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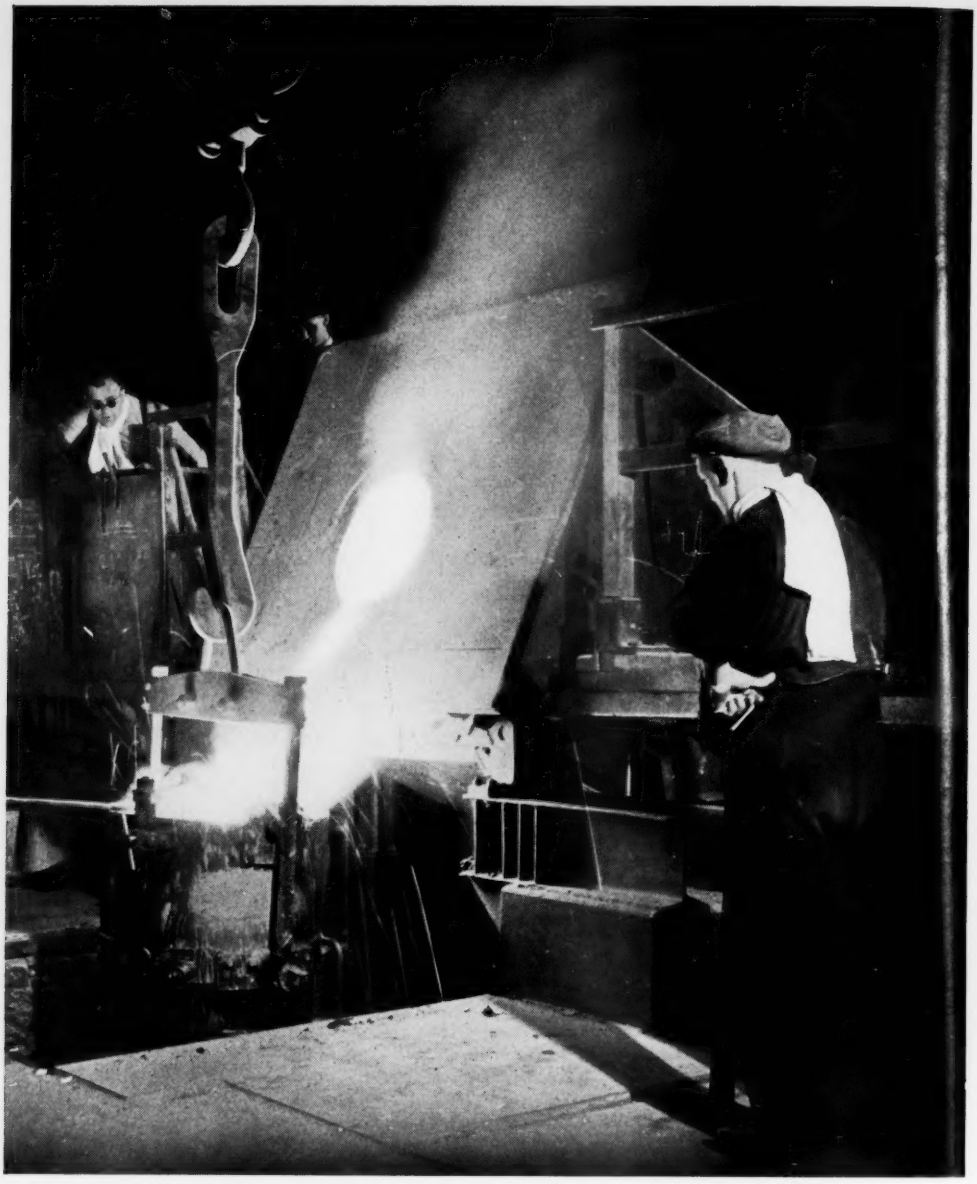

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GUSTAVUS HENRY LATHAM, J.P.

GUSTAVUS HENRY LATHAM, J.P., was born at Stoke-on-Trent in 1888. After three years with the London and North-Western Railway Company he joined the staff of L. D. Whitehead and Company in 1905, and in the following sixteen years he passed through all departments of the works to gain a thorough knowledge of all branches of production. During this period he was intimately concerned with the development of continuous rolling mills.

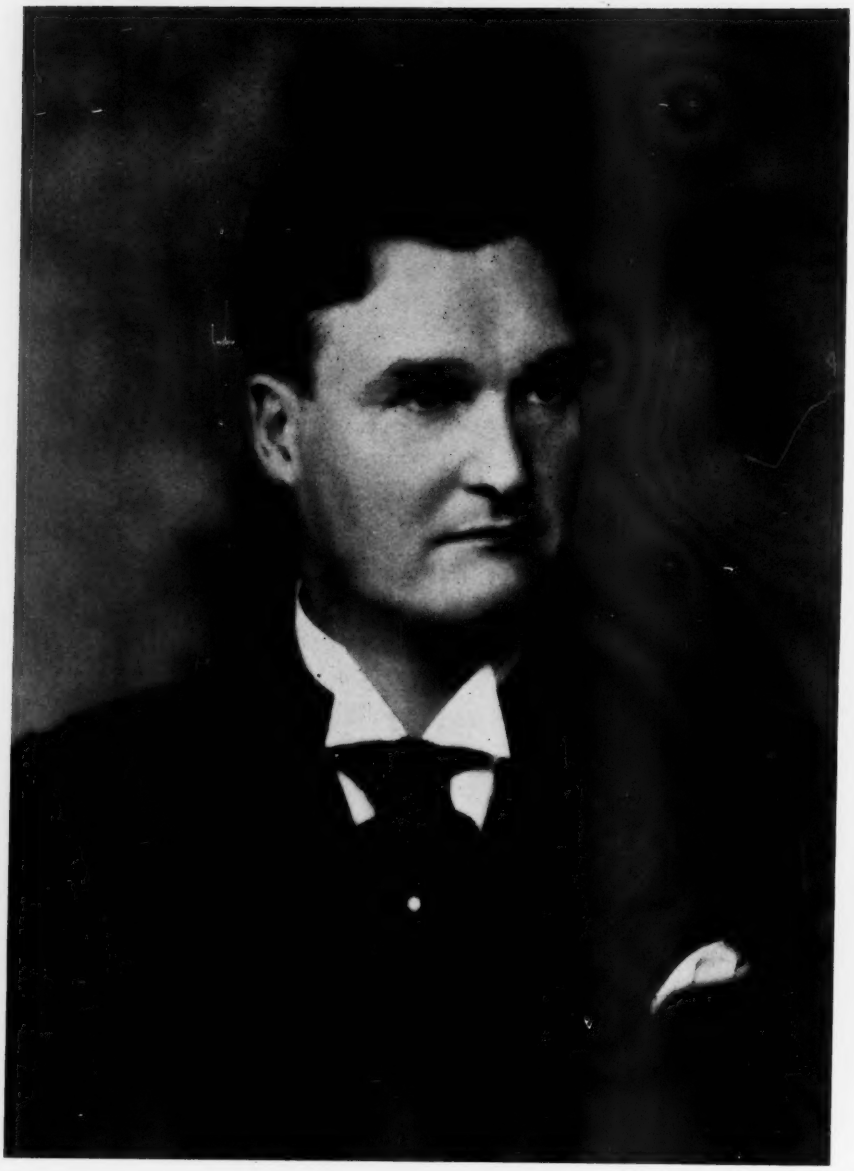
In 1921 Mr. Latham was appointed sales manager of the Whitehead Iron and Steel Company, Ltd., and ten years later he became general manager and Deputy Chairman. In 1938, following the death of Mr. L. D. Whitehead, he was appointed Chairman and Managing Director of the company, positions which he still occupies.

At the end of 1939, Mr. Latham was appointed by the Bank of England Control Committee to be Managing Director of Messrs. Richard Thomas and Company, Ltd., and he held this appointment until 1945, when the Committee ceased to function.

Mr. Latham has been a prominent figure in the iron and steel industry for many years. He is a Past-President of the British Iron and Steel Federation and is a member of the Executive Committee, Council, and Finance Committee, both of that body and of the British Iron and Steel Corporation, Ltd., of which he is a Director. He is also a member of various other committees of the Federation, and Chairman of Conference 13; he is also Chairman of the Executive, Management, and Standing Committees of the National Association for Rolled and Re-rolled Steel Products.

On the formation of the Finance Corporation for Industry, Ltd., Mr. Latham was appointed Technical Adviser for the whole of the steel industry, and in 1947 became a Member of the Iron and Steel Control Board.

He has always taken a keen interest in the technical and scientific side of the steel industry, and he was the Foundation President of the Newport and District Metallurgical Society. In 1947 he was appointed President of the British Iron and Steel Research Association. He became a Member of The Iron and Steel Institute in 1936, was elected a Member of Council in 1940 and a Vice-President in 1946.



Gustavus Henry Latham, J.P.
Vice-President

IRON AND STEEL
INSTITUTE PAPERS

PLANT DESIGN · MANUFACTURE · RESEARCH · DEVELOPMENT

Rapid Estimation of Slag Basicity

By W. A. Smith, B.Sc., A.R.T.C., J. Monaghan, B.Sc., A.I.M., and W. Hay

SYNOPSIS

Experiments have been carried out on the *p*H method of estimating slag basicity developed by Philbrook. Examination of all relevant factors failed to improve the accuracy beyond $\pm$ 0·3 *V* ratio, $CaO/(SiO_2 + P_2O_5)$.

A new method has been developed whereby the specific conductance of the water extract is measured and related to the *V* ratio of the slag. On homogeneous slags the *V* ratio can be estimated within $\pm$ 0·15 in a little over 20 min. The method is affected by the presence of undissolved lime in the sample, but the estimation is reliable when the sample is homogeneous.

Introduction

In the application of any system of slag control, the rapid assessment of the basicity of slags is of prime importance, and during the past few years considerable attention has been paid, both in this country, and in the United States, to the methods whereby the CaO/SiO_2 ratio, or the *V* ratio, $CaO/(SiO_2 + P_2O_5)$, may be rapidly estimated. Several systems of slag control are in use, or have been proposed, in which the slag basicity is estimated by such methods as :

(*a*) Chemical analysis
(*b*) Visual estimation from slag pancakes
(*c*) Examination of the colour of ground slags
(*d*) Petrographic examination of polished sections of slag.

In one of the more successful systems of slag control described by Fairley[1] in a recent paper, a combination of both chemical analysis and examination of pancakes is used for assessing slag basicity and controlling the feed additions to the furnace.

Although considerable success has been attained with such systems, there has still remained the need for a method for estimating the slag basicity with an accuracy which permits the additions of lime to the furnace to be calculated, and with a speed which avoids delay in making the additions.

The *p*H Method

A method of estimating slag basicity has been proposed by Philbrook, Jolly, and Henry[2] in which the faint alkalinity produced when a small portion of ground slag is shaken with water is measured by a *p*H meter. These workers established an empirical relationship between the basicity of the slag, as measured by the CaO/SiO_2 ratio, and the *p*H of the slag suspension. By this technique, details of which are given later, the CaO/SiO_2 ratio can be estimated in a little over 15 min from receipt of the slag in the laboratory, the result being accurate within $\pm$ 0·3 CaO/SiO_2 ratio in most cases, with occasional errors as high as $\pm$ 0·5 CaO/SiO_2 ratio.

As part of a programme for the development and application of a system of slag control, investigations were carried out on the *p*H method of estimating basicity. It is considered that to be of practical use as a basis for slag-control calculations, or as a check on pancake readings, an accuracy of at least $\pm$ 0·15 *V* is desirable, and the experiments were carried out to determine whether this degree of accuracy could be obtained by suitable modifications, or more rigid standardization of the procedure.

It was found, however, that such an accuracy is not inherent in the method and the investigations were then directed to finding a new procedure which would satisfy the requirements.

The Conductance Method

The close relationship between the specific conductance and the *p*H of solutions of electrolytes suggested that the conductivity of the water extract might also be related to slag basicity. Experiment proved that such is the case, and, in fact, a much closer degree of correlation exists between the conductivity of the water extract and the *V* ratio of the slag than had been found in the *p*H method.

The similarity to the *p*H technique is obvious, and further experiments showed that very similar techniques are required; the differences are mainly in detail right up to the final measurement. After the final technique had been established it was found possible to estimate the slag *V* ratio to within $\pm$ 0·15 *V* in most cases, the maximum error likely

Manuscript received 15th June, 1948.
Mr. Smith, Mr. Monaghan, and Mr. Hay are members of the staff of the Department of Research and Technical Development (Scottish Branch) of Stewarts and Lloyds, Ltd., Coatbridge, Lanarkshire.

to occur being 0·2 *V* ratio, an order of accuracy approaching that of rapid chemical analysis and comparing very favourably with other methods.

It is not perhaps obvious why the conductometric method should prove superior to the *p*H technique for estimating the basicity of the slag. Both are purely empirical methods based on physical measurements on the water extract of the slag, and are dependent on the slight solubility and ionization of slag constituents.

The conductance effect depends on the total number of ions formed due to this solubility and ionization, and is a measure of the ability of these ions to transmit electricity through the liquid, whereas the *p*H method is a measure of the concentration of the hydrogen ion only, which in turn is dependent upon the alkalinity of the water arising from the solution of the basic constituents.

There is no 'true' specific conductance for any particular slag *V* ratio, and, like the *p*H method, close adherence to procedure is necessary to obtain reproducible results.

The difficulties associated with the *p*H meter because of alkaline attack on the glass electrodes at high *p*H are not encountered with the platinized electrodes used on the conductivity bridge, nor is there any time lag in obtaining the reading.

It must be stated that the method suffers from the serious disadvantage that it can only be applied to homogeneous slags approaching equilibrium. The presence of undissolved lime in the sample raises the conductivity of the extract considerably, giving erroneous results.

INVESTIGATIONS ON THE *p*H METHOD

Type of Electrode

Using a normal glass electrode the measurement of $p\text{H} > 10$ is difficult because of the errors introduced through the solubility of the glass in highly alkaline solutions. This error may be reduced by standardizing with a buffer solution in the range 10–12 *p*H. An electrode recommended by the Cambridge Company for measuring $p\text{H} > 10$, known as the "Alki" electrode, has a high resistance to alkaline attack and is standardized with a 9·23 *p*H buffer solution. A comparison was made between the two types of electrode on a series of slag suspensions, the normal glass electrode being standardized by means of a 10·55 *p*H buffer solution. The results are given in Table I below, and it can be seen that there is a negligible difference in the readings obtained. Although it is possible to use an ordinary glass electrode for measuring the *p*H of slag suspensions, the Alki electrode was used in all subsequent measurements on account of its greater resistance to attack.

Factors Affecting the *p*H Measurement

In his original paper Philbrook[2] laid down the procedure for estimation of slag basicity by *p*H as follows:

> The slag sample is crushed to pass a — 40-mesh screen and metallic iron is removed. Approximately 0·20 g of the powdered sample, measured by means of a scoop, is shaken intermittently with 100 ml of water in a stoppered flask for 10 min. The *p*H meter is standardized by the usual procedure, the temperature of the slag suspension adjusted to 75–80° F, and the *p*H measured.

In arriving at this technique, Philbrook investigated several factors affecting the reproduction of *p*H readings and showed the most important to be:

1. Absorption of CO_2 from the atmosphere
2. Temperature of slag suspension when *p*H is measured
3. Particle size of slag powder used
4. Time of shaking before *p*H is measured
5. Weight of sample per given volume of water

CO_2 absorbed from the atmosphere will lower the *p*H of alkaline solutions, but Philbrook showed that this difficulty could be overcome by keeping the flask stoppered while shaking, and measuring the *p*H without delay. Freshly boiled distilled water may be used as an extra precaution.

It is clear also that the *p*H will be affected by the temperature of the slag suspension, and it is important to adjust this temperature to some standard value before measuring the *p*H.

Particle Size of Powder

Both the studies of Philbrook,[2] and the later work by Tenenbaum and Brown,[3] indicate that the *p*H of the slag suspension is greatly affected by the particle size of the slag powder. Philbrook found that the greatest contribution to the *p*H is made by particles of — 200 mesh, and concluded that if the sample is crushed so that it all passes a — 40-mesh screen there will be sufficient — 200-mesh powder to give reproducible results. Tenenbaum showed that the *p*H value varies with the percentage of — 200-mesh powder in the sample, the least variation being in the

Table I

EFFECT OF PARTICLE SIZE AND SHAKING TIME ON *p*H OF SLAG SUSPENSIONS

Type of Electrode	Particle Size	Slag *A* *V* ratio 1·73			Slag *B* *V* ratio 1·41			Slag *C* *V* ratio 1·35			Slag *D* *V* ratio 1·47		
		Shaking time in min (0·2 g per 100 ml of water)											
		5	10	15	5	10	15	5	10	15	5	10	15
Normal glass	— 40		10·64			10·59			10·16			10·36	
	— 80		10·84			10·81			10·43			10·50	
	—120		10·84			10·78			10·50			10·58	
Alki glass	— 40	10·35	10·62	10·56	10·45	10·60	10·58	10·13	10·18	10·20	10·28	10·34	10·32
	— 80	10·57	10·82	10·79	10·64	10·82	10·80	10·47	10·44	10·46	10·48	10·46	10·50
	—120	10·60	10·85	10·80	10·61	10·83	10·84	10·50	10·50	10·52	10·51	10·60	10·58

range 55–60%. This is best obtained by crushing the slag to pass a — 80-mesh screen.

Because of this difference in technique it was considered necessary to investigate the point further. Four slags were crushed to pass a — 40-, — 80-, and — 120-mesh screen in turn. A 0·2-g sample was taken at each stage, shaken with 100 ml of distilled water for 10 min, and the *p*H of the suspension measured. The results given in Table I indicate that there is very little difference between the — 80- and — 120-mesh samples, but the — 40 mesh gives results much lower than either. Consequently it was decided to adopt the following technique for sampling: the entire pancake is roughly crushed to pea size, quartered, and a portion ground to pass a — 120-mesh screen. The final grinding may be done quickly and efficiently by a small cone-type mill.

Time of Shaking

A check was also made on the effect of the time of shaking by measuring the *p*H of 0·2 g/100 ml of water after 5, 10, and 15 min shaking, using the same samples as previously. These results, also quoted in Table I, confirm Philbrook's conclusion[2] that the *p*H value is constant after 10 min shaking.

Weight of Slag per 100 ml of Water

The effect of weight of slag used per 100 ml of water was determined on 4 slag samples, the volume of water used and the shaking time being constant. These results are plotted in Fig. 1.

It is evident that a constant *p*H is not obtained until the sample weight is in the range 0·5–0·8 g per 100 ml of water, but reproducible results can be obtained with a sample weight of 0·5 g if the amount is weighed accurately.

This result is in contrast to the finding of Philbrook[2] that the maximum *p*H of the suspension is developed with a sample weight of 0·20 g per 100 ml of water.

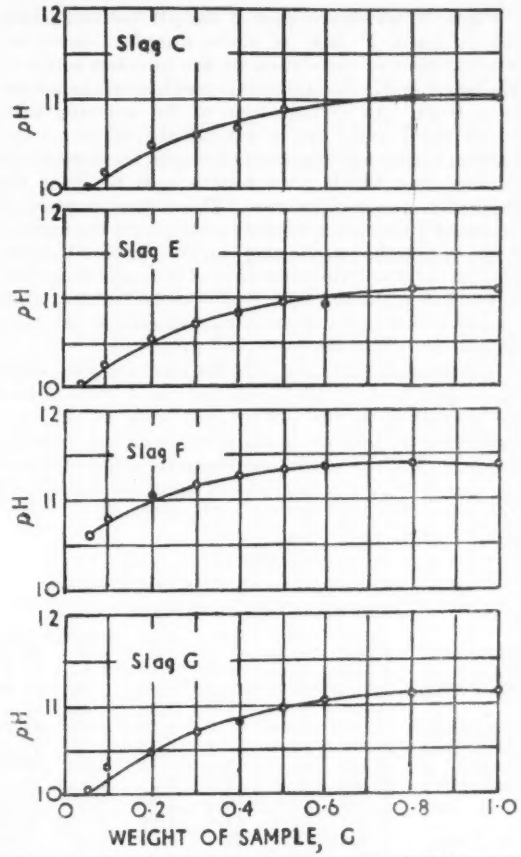


Fig. 1—Effect of sample weight on *p*H of slag water extracts. Shaking time, 10 min

Relationship between *p*H Value and Slag *V* Ratio

Having established the details of procedure, the *p*H values of two series of slag suspensions were determined, the first series using a sample weight of 0·2 g per 100 ml of water as recommended by Philbrook, and the second series using 0·5 g/100 ml. For either weight of sample the technique adopted was as follows:

> The slag, crushed to pass a — 120-mesh screen, was weighed into a stoppered flask, 100 ml of CO_2-free distilled water were added and shaken for 10 min, the temperature of the suspension was adjusted to 25° C, and the *p*H was measured.

Using this technique, the *p*H values of suspensions of 47 slags were determined with 0·2 g/100 ml sample weight, and related to the slag basicities. The slags contained up to 10% of P_2O_5 and this was corrected for by comparing the *p*H measurement with the *V* ratio $CaO/(SiO_2 + P_2O_5)$ of the slags calculated from the chemical analysis. These results are given in Table II.

The *p*H value is plotted against the log *V* ratio in Fig. 2; the line shown is the calculated regression line. The scatter band is very broad and the degree of correlation low—the calculated correlation coefficient is + 0·77. The relationship is quite definite, but six of the points are more than 0·5 *V* from the regression line. No improvement is found on plotting *p*H against the expressions, log $(CaO - 1{\cdot}18\ P_2O_5)/SiO_2$, or log $CaO/(SiO_2 + 0{\cdot}634\ P_2O_5)$.

In the second series the *p*H values were measured on slag suspensions of 73 slags using a 0·5-g sample weight. These data are given in Table III.

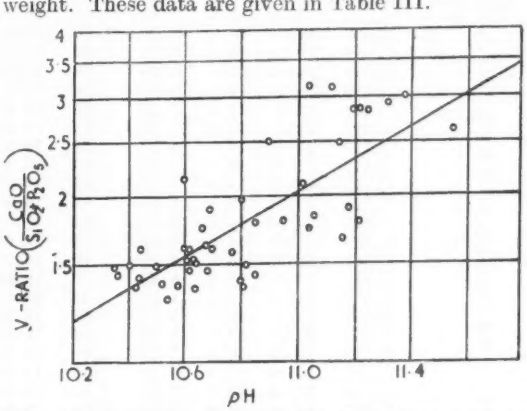


Fig. 2—Relationship between log *V* ratio and *p*H of slag water extracts (0·2 g/100 ml)

Figure 3, which is a plot of the *p*H values against the analysed *V* ratio of these samples, shows an improvement in correlation on the previous series of 47 slags (Fig. 2) ; the correlation coefficient is increased to + 0·85. As an indication of the accuracy with which the *V* ratio can be estimated, and as a comparison between the two sets of results, the percentage of the slags which come within ± 0·15 *V* of the regression line is of interest. This is the standard of accuracy which is considered necessary for the method to be of practical use in slag control. Table IV shows that in the first series only 32% of the slags are within ± 0·15 *V* ratio and 60% in the second series, while nearly 1 in 8 and 1 in 20 slags respectively are more than 0·5 *V* from the regression line.

It is evident that even with the modifications which have been made to the technique, the *V* ratio cannot be estimated by the *p*H method with an accuracy sufficient to provide a reliable check on pancake readings, and the conclusion drawn from these investigations is that the standard of accuracy desired for slag control is not inherent in the method.

Table II

VALUES OF *p*H FOR AQUEOUS SLAG SUSPENSIONS (0·2g/100 ml)

Slag		Slag	SiO_2	CaO	P_2O_5	*V* Ratio	*p*H
Melt	Slag	J.70B	31·33	45·1	1·50	1·37	10·81
Tap	Slag	J.71B	17·82	54·9	3·12	2·62	11·57
Melt	Slag	J.122B	19·24	48·3	3·86	2·10	11·04
,,	,,	C.204	21·54	47·2	7·38	1·64	10·44
Tap	Slag	C.204	13·94	55·7	5·50	2·87	11·20
Melt	Slag	C.245	26·16	45·7	4·13	1·51	10·40
Tap	Slag	C.245	16·12	53·2	5·15	2·50	10·92
Melt	Slag	C.249	26·2	42·0	4·72	1·36	10·64
,,	,,	C.259	19·4	46·0	7·95	1·68	11·16
,,	,,	C.264	24·0	44·6	5·91	1·49	10·35
Tap	Slag	C.264	14·7	46·7	4·03	2·49	11·15
Melt	Slag	C.268	24·18	41·6	7·24	1·37	10·42
Tap	Slag	C.268	11·76	53·5	5·92	3·02	11·38
Melt	Slag	C.270	20·18	45·0	8·16	1·58	10·77
Tap	Slag	C.270	12·74	54·7	5·94	2·93	11·32
Melt	Slag	C.297	21·38	52·2	4·97	1·98	10·80
Tap	Slag	C.297	15·12	59·8	3·94	3·14	11·04
Melt	Slag	C.313	20·4	55·8	5·52	2·15	10·60
Tap	Slag	C.313	14·7	58·7	4·13	3·12	11·12
Melt	Slag	C.317	22·2	52·8	5·90	1·88	10·69
Tap	Slag	C.317	14·82	56·4	4·79	2·88	11·22
Melt	Slag	C.320	22·96	48·0	6·56	1·63	10·68
Tap	Slag	C.320	14·5	57·0	5·40	2·86	11·26
Melt	Slag	C.366	19·74	45·5	9·34	1·53	10·61
,,	,,	C.367	19·06	49·9	7·76	1·80	10·95
,,	,,	C.368	23·56	44·2	5·95	1·50	10·82
,,	,,	C.369	20·46	27·8	6·45	1·78	10·85
,,	,,	C.370	20·6	44·1	6·95	1·60	10·61
,,	,,	C.371	23·0	44·2	7·22	1·46	10·68
,,	,,	C.372	22·3	44·3	7·53	1·49	10·50
,,	,,	C.373	27·56	42·7	4·17	1·38	10·52
,,	,,	C.374	25·54	44·7	5·87	1·47	10·62
,,	,,	C.375	23·12	43·9	7·57	1·43	10·44
,,	,,	C.376	23·96	45·5	6·02	1·54	10·63
,,	,,	C.379	24·80	44·2	5·87	1·44	10·85
,,	,,	C.380	21·88	50·8	6·25	1·75	11·04
,,	,,	C.381	26·58	44·8	4·09	1·40	10·80
,,	,,	C.382	21·0	48·2	6·45	1·75	10·66
,,	,,	C.390	21·12	42·2	9·80	1·37	10·58
,,	,,	C.391	17·2	48·8	8·72	1·89	11·18
,,	,,	C.392	21·86	43·0	8·00	1·44	10·36
,,	,,	C.393	21·39	45·8	7·18	1·60	10·70
,,	,,	C.395	16·50	50·1	10·81	1·84	11·06
,,	,,	C.396	19·84	48·9	7·33	1·80	11·22
,,	,,	C.397	24·58	39·9	6·10	1·30	10·54
,,	,,	C.398	23·32	47·5	6·02	1·62	10·60
,,	,,	C.401	20·44	43·0	7·64	1·53	10·64

INVESTIGATIONS ON THE CONDUCTANCE METHOD

PROCEDURE

When it became clear that the *p*H method could not be developed to the standard of accuracy required, the investigations were directed towards finding an alternative procedure—the conductivity of a water extract.

Table III

VALUES OF *p*H FOR AQUEOUS SLAG SUSPENSIONS (0·5g/100 ml)

Slag		Slag	SiO_2	CaO	P_2O_5	*V* Ratio	*p*H
Melt	Slag	J.70B	31·30	45·1	1·50	1·37	11·05
Tap	Slag	J.71B	17·82	54·9	3·12	2·62	11·54
Melt	Slag	J.122B	19·24	48·3	3·86	2·10	11·14
,,	,,	C.204	21·54	47·2	7·38	1·64	10·82
Tap	Slag	C.204	13·94	55·7	5·50	2·87	11·64
Melt	Slag	C.245	26·16	45·7	4·13	1·51	10·90
Tap	Slag	C.245	16·12	53·2	5·15	2·50	11·32
Melt	Slag	C.249	26·20	42·0	4·72	1·36	11·17
,,	,,	C.259	19·40	46·0	7·95	1·68	11·48
,,	,,	C.264	24·0	44·6	5·91	1·49	11·08
Tap	Slag	C.264	14·70	46·7	4·03	2·49	11·51
Melt	Slag	C.268	24·18	41·6	7·24	1·37	10·72
Tap	Slag	C.268	11·76	63·5	5·92	3·02	11·76
Melt	Slag	C.270	20·18	45·0	8·16	1·58	11·19
Tap	Slag	C.270	12·74	54·7	5·94	2·93	11·68
Melt	Slag	C.297	21·38	52·2	4·97	1·98	11·28
Tap	Slag	C.297	15·12	59·8	3·94	3·14	11·60
Melt	Slag	C.313	20·40	55·8	5·52	2·15	11·16
Tap	Slag	C.313	14·70	58·7	4·13	3·12	11·66
,,	,,	C.317	14·82	56·4	4·79	2·88	11·76
Melt	Slag	C.320	22·96	48·0	6·56	1·63	11·18
Tap	Slag	C.320	14·50	57·0	5·40	2·86	11·78
Melt	Slag	C.317	22·20	52·8	5·90	1·88	11·38
,,	,,	C.366	19·74	45·5	9·34	1·53	11·24
,,	,,	C.367	19·06	49·9	7·76	1·80	11·56
,,	,,	C.368	23·56	44·2	5·95	1·50	11·28
,,	,,	C.369	20·46	47·8	6·45	1·78	11·09
,,	,,	C.370	20·60	44·1	6·95	1·60	11·15
,,	,,	C.371	23·0	44·2	7·22	1·46	11·10
,,	,,	C.372	22·30	44·3	7·53	1·49	10·98
,,	,,	C.373	27·56	42·7	4·17	1·38	10·94
,,	,,	C.374	25·54	44·7	5·87	1·47	11·20
,,	,,	C.375	23·12	43·9	9·57	1·43	11·06
Tap	Slag	C.427	14·08	54·6	4·09	2·88	12·04
Melt	Slag	C.428	23·0	41·3	5·79	1·43	11·27
Tap	Slag	C.428	13·92	52·5	4·71	2·82	11·98
Melt	Slag	C.429	21·28	41·8	7·95	1·43	11·38
,,	,,	C.376	23·96	45·5	6·02	1·54	11·21
,,	,,	C.379	24·80	44·2	5·87	1·44	11·18
,,	,,	C.380	21·88	50·8	6·25	1·75	11·36
,,	,,	C.381	26·58	44·8	4·09	1·40	11·25
,,	,,	C.382	21·0	48·2	6·45	1·75	11·21
,,	,,	C.390	21·12	42·2	9·80	1·37	10·92
,,	,,	C.391	17·20	48·8	8·72	1·89	11·44
,,	,,	C.392	21·86	43·0	8·00	1·44	10·95
,,	,,	C.397	24·58	39·9	6·10	1·30	11·00
,,	,,	C.398	23·32	47·5	6·02	1·62	11·13
,,	,,	C.393	21·39	45·8	7·18	1·60	11·26
,,	,,	C.400	22·48	49·0	5·17	1·77	11·21
,,	,,	C.395	16·50	50·1	10·81	1·84	11·30
,,	,,	C.396	19·84	48·9	7·33	1·80	11·66
,,	,,	C.401	20·44	43·0	7·64	1·53	11·14
,,	,,	C.419	21·60	45·3	5·40	1·68	11·06
Tap	Slag	C.419	14·7	53·0	4·63	2·74	11·78
Melt	Slag	C.420	24·56	41·3	5·32	1·38	11·32
Tap	Slag	C.420	14·24	53·8	3·94	2·96	11·84
Melt	Slag	C.421	20·68	39·1	6·95	1·42	11·08
Tap	Slag	C.421	10·60	52·6	6·71	3·04	11·81
Melt	Slag	C.422	16·24	49·5	8·80	1·98	11·59
Tap	Slag	C.422	10·24	50·26	5·79	3·14	12·06
Melt	Slag	C.423	20·32	47·2	6·95	1·74	11·18
Tap	Slag	C.423	12·28	53·3	5·33	3·02	12·00
,,	,,	C.429	11·0	53·5	6·39	3·26	12·08
,,	,,	C.432	11·56	52·5	4·79	3·21	12·13
,,	,,	C.433	12·68	50·7	6·33	2·67	11·96

Table IV

COMPARISON OF ACCURACY OF *p*H METHODS

Series	Sample Wt, g/100 ml	Correl. Coeff. V	Deviation from Regression ±0·15 V	±0·3 V	±0·5 V
First (Fig. 2)	0·2 g.	+0·77	32%	66%	87%
Second (Fig. 3)	0·5 g.	+0·85	60%	75%	95%

Conductivity measurements were made on water extracts of several ground slags and quite a close relationship was found between the specific conductance of the extract and the *V* ratio of the slag, calculated from the analysis. The extent of solution and hydrolysis of the slag constituents is quite small, so that the specific conductance of the extract is normally of the very low order of $0 \cdot 5/5 \times 10^{-4}$. The measurement is easily carried out on one of several types of conductivity bridge commercially obtainable.

It was thought that the conductivity of the water used might seriously affect the readings obtained, and the use of specially prepared conductivity water, which has a conductance of the order 1×10^{-6}, was considered. However, CO_2-free distilled water was found to give a negligible reading on the particular conductivity bridge used, and had a conductivity sufficiently low to have no bearing on the final conductivity of the water extract. The use of specially prepared conductivity water is therefore not considered necessary, the only precaution that need be taken is that the distilled water should be freshly prepared and kept in a stoppered flask.

The technique first adopted for preparing the water extract was similar to that used when measuring the *p*H, but one very important modification was necessary. The presence of slag particles suspended in the liquid, or adhering to the electrodes, seriously interferes with the measurement, and it is necessary to filter off the slag particles and measure the conductivity on the clear filtrate. This precaution is not necessary in the *p*H method, as the presence of the suspended particles has no effect on this measurement.

The first results obtained were on water extracts of 41 slags using the following technique :

0·25 g of slag, crushed to pass a — 120-mesh screen, were weighed, placed in a conical flask with 100 ml of CO_2-free water, the flask stoppered and shaken for 15 min, and the suspension filtered through a close 11-cm filter paper (Whatmans No. 42). The conductivity was measured on the clear filtrate.

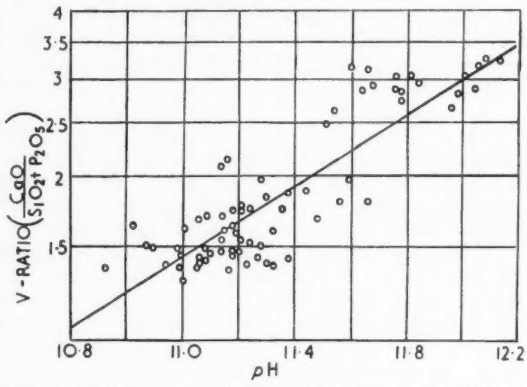


Fig. 3—Relationship between log *V* ratio and *p*H of slag water extracts (0·5 g/100 ml)

The results obtained on these slags are quoted in Table V.

On plotting the specific conductance against the *V* ratio calculated from the analysis, a curve is obtained

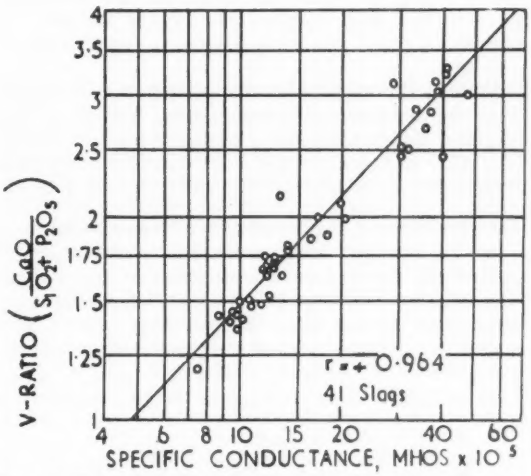


Fig. 4—Relationship between log *V* ratio and log specific conductance of water extracts (0·25 g/100 ml)

Table V

SET 1. CONDUCTIVITIES OF AQUEOUS EXTRACTS OF SLAGS (0·25g/100 ml)

Slag	SiO_2	CaO	P_2O_5	*V* Ratio	Spec. Conduct., mhos × 10^5
Melt C.409	18·48	48·6	7·64	1·87	18·20
,, C.423	20·30	47·2	6·95	1·74	12·40
Tap C.423	12·28	53·3	5·33	3·02	38·50
Melt C.422	18·24	49·3	8·80	1·88	20·80
Tap C.422	10·24	50·26	5·79	5·14	38·20
Melt C.429	21·28	41·8	7·95	1·43	11·00
Tap C.429	11·00	53·5	6·39	3·26	41·30
Melt C.428	23·00	41·3	5·79	1·43	8·90
Tap C.428	13·92	52·5	4·71	2·82	36·80
Melt C.427	25·60	44·8	2·93	1·57	12·60
Tap C.427	14·08	54·6	4·09	2·68	33·60
Melt C.433	23·04	43·1	6·79	1·44	9·78
Tap C.433	12·68	50·7	6·33	2·67	35·70
Melt C.432	22·84	41·1	6·25	1·41	10·68
Tap C.432	11·56	52·5	4·79	3·21	41.00
Melt C.431	24·60	43·6	7·26	1·37	10·14
Tap C.431	13·02	56·0	5·71	2·99	47·60
Melt C.438	24·72	37·0	6·71	1·18	7·80
Tap C.438	14·64	50·8	5·71	2·50	31·70
Melt C.437	18·04	41·3	6·33	1·70	12·25
Tap C.437	12·08	51·3	4·40	3·12	28·70
Melt C.436	22·70	44·7	6·40	1·54	11·90
,, C.454	18·60	49·7	5·36	2·08	19·80
,, C.453	20·00	45·5	4·59	1·85	16·30
,, C.452	16·68	49·2	2·93	2·51	30·30
,, C.451	24·04	42·2	3·94	1·51	10·84
,, C.463	18·60	46·2	4·86	1·97	17·15
,, C.462	19·08	44·0	7·18	1·68	11·90
,, C.461	22·80	41·0	5·79	1·43	10·00
,, C.460	17·48	54·0	7·87	2·13	13·30
,, C.467	21·72	46·6	5·87	1·69	12·80
Tap C.467	15·16	52·4	6·18	2·46	30·10
Melt C.466	21·40	41·4	3·32	1·72	12·90
Tap C.466	18·08	51·6	2·93	2·44	40·60
Melt C.471	21·70	46·0	6·25	1·64	13·60
,, C.473	23·54	41·2	5·67	1·41	9·50
,, C.380	31·88	50·8	6·25	1·75	14·00
,, C.468	18·70	38·3	4·63	1·64	12·20
,, C.484	24.06	44.6	5.63	1·50	10·50
,, C.400	22·48	49·0	5·17	1·77	14·00
,, C.406	19·02	45·6	7·01	1·75	12·90

which suggests a rather complex relationship. The relationship can be best represented by a logarithmic law, and on plotting the log specific conductance against log V ratio a straight line is obtained as shown in Fig. 4. With the coefficient of correlation as $+0{\cdot}964$, and despite the limited number of samples, a very promising degree of correlation is achieved.

Four of the samples show rather large deviations from the calculated regression line, but 33 of the samples, representing 83% of the total, are within $\pm 0{\cdot}15$ V of the regression line. This compares with 60% within $\pm 0{\cdot}15$ V for the pH results in Fig. 3.

Factors Affecting the Conductivity Measurement

In an attempt to improve the correlation and reduce the scatter, an investigation of the various factors influencing the reproduction of the conductance value was carried out. The following factors were found to have the greatest influence on the specific conductance of the water extract :

(*a*) Temperature of aqueous extract at time of measuring the conductivity
(*b*) Effect of filter papers
(*c*) Particle size of slag powder used
(*d*) Weight of slag per 100 ml of water
(*e*) Time of shaking before measurement of the conductivity
(*f*) The effect of undissolved CaO.

Effect of Temperature

According to Findlay,[5] the conductivity changes with variations in temperature in a rather complex manner, as it is dependent on the change in ionization of the electrolyte, and on the changes in dielectric constant and viscosity of the water with temperature. At ordinary temperatures the conductivity increases by about 2% per ° C. It is recommended, therefore, that the temperature of the water extract be adjusted to a standard value before measuring the conductivity. In the experimental work a standard temperature of 18° C was adopted.

Effect of Filter Papers

The necessity for filtering off the suspended slag particles before measuring the conductivity has already been pointed out. As most types of filter paper are acid-washed during manufacture it is probable that some acid is retained in the paper, and this can affect the conductivity reading. A comparison made between the specific conductances of extracts of several slags using dry and washed papers, has shown that the specific conductance is enhanced if the paper is thoroughly washed with CO_2-free distilled water and allowed to drain before use. The effect is quite small, corresponding to a rise of about 0·05–0·1 in the V ratio, but its extent is liable to vary from one batch of filter papers to another, and consequently it was considered advisable to use washed papers for all subsequent work. At the same time, the Whatmans No. 42 papers were replaced by Greens No. 803, as the latter have a higher filtration rate while still giving a clear filtrate.

Effect of Particle Size

The influence of particle size of the slag powder in the pH method has already been discussed. The size of particles might be expected to have a similar effect on the conductivities of water extracts. Tests on several slags have shown that higher conductivity measurements are obtained on crushing the slag to pass a — 200-mesh screen than those obtained if the slag is passed through a — 120-mesh screen. Data derived for one slag are as follows :

Slag N. V *ratio* 1·88 (0·2 *g*/100 *ml*)

Crushed to pass :	Spec. Conduct., mhos × 10^6
— 120 mesh	13·8
— 200 mesh	16·95

In choosing a suitable particle size, consideration must be given to the effect of variations in sampling technique on the size analysis of the final sample, the time required for preparation of the sample, and the possible effect of the particle size on the shaking time required to reach a constant conductivity value.

If the sample is crushed to pass only, say, — 40 mesh, the proportion of under — 200-mesh powder in the sample can vary considerably with the method of preparation, and similarly, excessive amounts of very fine powder will result if it is attempted to pass all the sample through a — 200-mesh screen. Most consistent results are likely to be achieved by mechanical crushing to about — 100 or — 120 mesh. The preparation of a sample of — 200 mesh requires considerable time in grinding, but the slag can be crushed to — 120 mesh in 2 or 3 min.

It will be shown later that the time of shaking required to give a constant conductivity is as long on the — 200-mesh as on the — 120-mesh samples.

There is no advantage therefore, in using very fine particle size, and in consequence all subsequent samples were prepared to pass a — 120-mesh screen, using the same technique as previously used and described for the pH method. A screen size of — 100 mesh is probably just as satisfactory as the — 120 mesh.

Effect of Weight of Slag in a Given Volume of Water, and Shaking Time

The early investigations showed that the specific conductance of the water extract is very dependent on the weight of slag per 100 ml of water and on the time of shaking before making the estimation. To investigate the effect of these factors, the specific conductances of water extracts of four slags of different V ratios were determined for various sample weights and shaking times. The samples were crushed to — 120 mesh and in each case 100 ml of water were used.

Sample Weight—Figure 5 shows the effect of weight of slag per 100 ml of water on the specific conductance of the extracts, each curve representing a different shaking time. The conductivity rises almost proportionally to the sample weight in the range 0·2–1 g/100 ml, when the shaking time is about 20–25 min, but with only 5 or 10 min shaking the graph curves off, so that at 1 g per 100 ml of water the conductivities measured after 5 min are much lower than the maximum possible after 25–30 min shaking. With the two more basic slags with V ratio 3·24, Fig. 5 (*a*), and V ratio 2·29, Fig. 5 (*c*), the relationship is not strictly linear, even at the maximum

shaking time. It is evident that up to a concentration of 1 g per 100 ml of water, in contrast to the *p*H, the specific conductance does not reach constancy with rise in sample weight, though there must be some limit corresponding to a saturated solution of the slag constituents in water.

Shaking Time—From the data plotted in Fig. 6 it can be seen that the conductivity is constant after 15 min shaking on the 0·2-g samples for three of the slags, but on slag *H*, with 3·24 *V* ratio, nearly 20 min are needed to reach a constant figure. With increasing sample weight the time to reach a stable conductivity rises, so that at 1 g of slag per 100 ml of water at least 20–25 min are needed to reach equilibrium. For the method to be of practical use it is essential for the shaking time to be as short as possible. If, however,

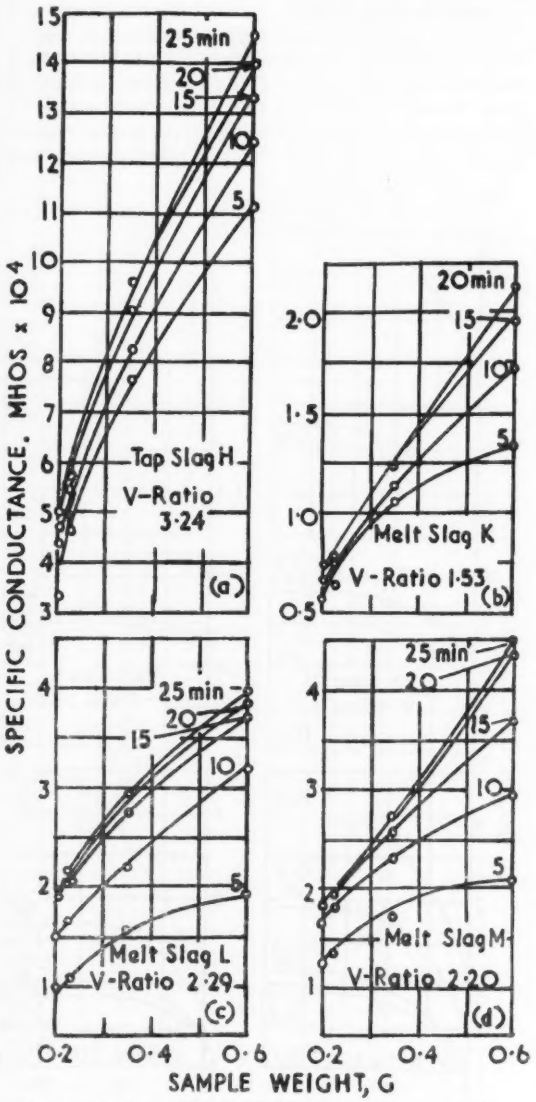


Fig. 5—Effect of sample weight on specific conductance of water extracts

Table VI

EFFECT OF SHAKING TIME ON THE CONDUCTIVITY FOR −200-MESH SAMPLE

Slag *N*. *V* ratio 1·88

Shaking Time, min	Conductivity, mhos × 10^5
5	11·7
10	15·4
15	16·95
20	16·95

the time is so short that the conductivity measurement has not reached constancy, any variation in filtration time will affect the conductivity value of the extract, and will interfere with the reproducibility of the observations. Furthermore, the measurements will, on the whole, be lower, and this effect will be more pronounced at high *V* ratios, so that the relationship between the logarithm of the specific conductance and the logarithm of the *V* ratio of the slag will no longer be linear. This probably accounts for some of the scatter at the higher *V* ratios found in the first set of results (Fig. 4).

With 15 min shaking, the specific conductance is not seriously affected by small delays before taking the measurement, provided that the sample weight is 0·20 g per 100 ml of water or lower.

In an attempt to reduce further the shaking time needed for equilibrium, a slag sample was crushed to pass a −200-mesh screen and the specific conductances were determined on water extracts of 0·2 g of slag per 100 ml of water after various shaking times; the data collected are given in Table VI which shows that the conductivity is not constant until nearly 15 min shaking time.

From these investigations it appears that the best combination of sample weight and shaking time is given by 0·2 g per 100 ml of water and 15 min shaking before filtration, and in consequence these conditions were adopted for determining the final calibration curve.

Effect of Undissolved Lime

Some experiments were carried out to determine the extent of the error introduced into the estimation by the presence of undissolved lime in the sample. In one of these tests a sample of a well-digested slag was drawn immediately prior to an addition of 8 cwt of burnt lime, and a further sample taken 7½ min after the addition. The latter sample was free from large lumps of lime, but on fracture numerous small particles of CaO could be seen dispersed through the pancake. The two samples were carefully analysed and a conductivity estimation of *V* ratio made on each. The data collected are given in Table VII.

The conductance estimate on the digested sample is only 0·08 *V* from the analysed *V* ratio, whereas after the additions it is 0·9 *V* higher. Therefore CaO,

Table VII

EFFECT OF UNDISSOLVED CaO ON CONDUCTANCE MEASUREMENT

	Slag Analyses CaO	SiO_2	P_2O_5	Calculated *V* ratio	Conductance *V* ratio
Before addn. well digested	41·8	21·76	5·56	1·53	1·45
7½ min. after addn. of 8 cwt B lime	45·7	19·24	5·40	1·85	2·75

which has not been taken into solution in the slag, has an effect on the conductivity of the extract far out of proportion to its concentration in the sample, and in consequence the method is not recommended for samples drawn a short time after additions of lime to the furnace, or on samples in which the presence of undissolved CaO may be discovered by examination.

Effect of Age of Slag

Estimations of the specific conductance of water extracts on slags up to two weeks old have shown no reduction in the conductivity measurement, provided the sample has been kept in good condition. This is useful, as the conductance method may be used to check the *V* ratio some time after the original estimation, should this be necessary. Nevertheless, it is advisable, when preparing a calibration graph for estimating *V* ratio by the method, to use only freshly prepared samples.

Final Technique for Conductance Method

Based upon the investigations of the factors affecting the specific conductance of the extract, a final technique has been drawn up for conductance estimations, and used to prepare a calibration graph for slag control. The procedure is as follows :

The slag pancake is broken down, mixed, and further crushed on an iron plate. The sample is then quartered and a portion ground by passing it twice through a small grinding mill, which enables it to pass completely a — 120-mesh screen. Metallic iron is removed by a magnet.

A sample of 0·20 g is weighed accurately and transferred to a 250-ml conical flask with 100 ml of CO_2-free distilled water ; the flask is stoppered and shaken intermittently for 15 min. An 11-cm Greens No. 803 filter paper is placed in a funnel, washed thoroughly with the CO_2-free distilled water, and allowed to drain. The slag suspension is filtered through the prepared paper, the first few millilitres being used to rinse the cell and electrodes, and finally enough of the filtrate collected to cover the electrodes completely ; normally 25–30 ml will be sufficient. The temperature of the extract is adjusted to 18° C and the conductivity measured on the appropriate scale of the conductivity bridge. The specific conductance of the extract is obtained by multiplying the meter reading by the cell constant. In using the method for estimating slag basicity, the *V* ratio is obtained from the calibration graph.

The following points should be noted in connection with this method :

(1) The distilled water should be freshly boiled and kept in a stoppered flask to maintain it CO_2-free, and should give no appreciable reading on the lowest scale of the conductivity bridge. The temperature of the water should be maintained at approximately 18° C.

(2) The electrodes should be immersed in distilled water when not in use. At regular intervals they should be cleaned by immersion in chromic-acid/sulphuric-acid mixture. After a time they may require replatinizing.

(3) The cell constant should be checked frequently by measuring the conductivity of a standard solution of potassium chloride (0·7463 g of KCl) dissolved in 1000 g of water having a true specific conductance of $1 \cdot 2202 \times 10^{-3}$ mhos at 18° C.

(4) No attempt should be made to check the conductivity observation on the last portion of the filtrate on an estimation, as much higher results will be obtained. This is because the slag, in the last portion of the suspensions to be filtered, is at a much higher concentration in the water, and in consequence, on filtering, the extract has a much higher conductivity. If it should be necessary to check any conductivity measurement, a fresh estimation should be made.

The Relationship between the Specific Conductance and the Slag *V* ratio

Having completed the details of technique, a fresh series of slags was examined in order to draw up a calibration graph by which the conductance method could be used to estimate the *V* ratio of the slag for production slag control. Conductivity measurements were made on extracts of 100 slags taken from normal

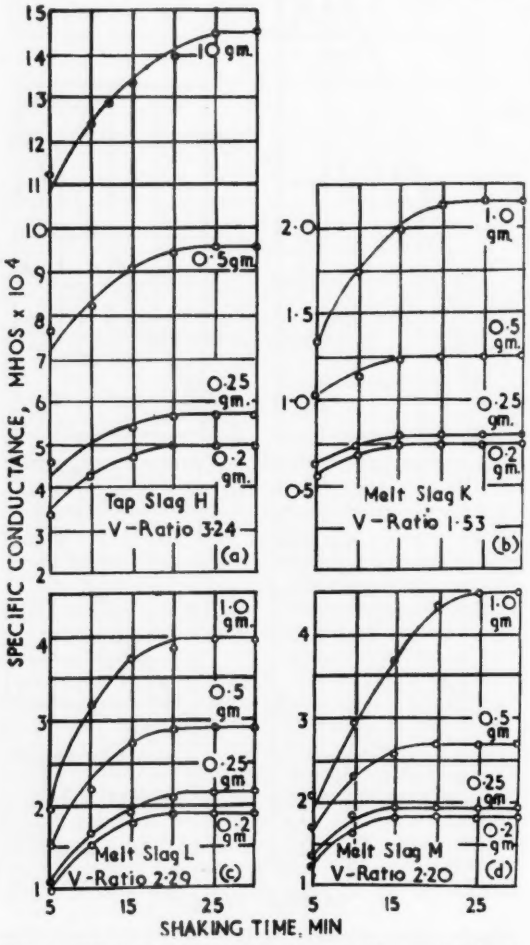


Fig. 6—Effect of shaking time on specific conductance

production, the samples being drawn at a period when the slag was homogeneous and free from undissolved lime. The importance of this precaution has already been discussed.

The V ratios were derived from very careful analyses of the slags for CaO, SiO_2, and P_2O_5, and are quoted, together with the specific conductance observations, in Table VIII.

In Fig. 8 the data have been plotted on logarithmic ordinates, for with this type of graph a straight line is obtained, showing that there is a linear relationship between the logarithms of the two measurements. The line of regression drawn has been calculated by the method of least squares, its equation being :

$$\text{Log } V = -0{\cdot}1597 + 0{\cdot}3793\ (5 + \log K),$$

where V is the ratio $CaO/(SiO_2 + P_2O_5)$ and K is the true specific conductance of the water extract in mhos.

The line is the regression of log V ratio on log conductance as it is required to estimate the V ratio from the specific conductance. The coefficient of correlation is very high ($+ 0{\cdot}992$) and an examination of the graph has shown that 82% of the points lie within $\pm 0{\cdot}1$ V, and 95% within $\pm 0{\cdot}15$ V of the regression line. None of the samples is more than $0{\cdot}2$ V ratio from the line. Comparison should be made with the data for the pH method (Table IV and Fig. 3).

In this particular series of samples the lowest V ratio was $1{\cdot}31$ V, but in later work this graph has been used to estimate successfully V ratios down to almost $1{\cdot}0$.

Provided that the sample is truly homogeneous, the method is almost as accurate for estimating the V ratio in the range $3{\cdot}0$–$4{\cdot}0$ V as it is for lower basicity, though there is, of course, a greater risk of heterogeneity with the higher V ratios. No samples have been examined with V ratios greater than $4{\cdot}0$ V, for these slags are very rarely obtained with normal furnace operation.

Under routine conditions, and comparing with the

Table VIII

SET II. CONDUCTIVITIES OF AQUEOUS EXTRACTS OF SLAGS (0·20 g/100 ml)

Slag		SiO_2	CaO	P_2O_5	V Ratio	Spec. Conduct., mhos × 10^4
Melt	C.723	18·38	43·2	4·73	1·87	13·65
Melt	C.724	16·52	49·8	5·17	2·30	25·20
Tap	C.726	13·80	51·6	3·35	3·01	52·50
Melt	C.727	19·30	46·3	4·43	1·95	15·75
Tap	C.727	12·52	53·6	3·13	3·42	57·40
Melt	C.728	17·04	49·8	5·47	2·22	21·35
Tap	C.728	10·52	51·8	3·86	3·60	75·25
Melt	C.730	25·00	41·2	3·47	1·45	7·70
Tap	C.730	14·48	53·2	4·17	2·85	44·10
Melt	C.731	12·98	51·5	5·55	2·78	43·40
Tap	C.731	11·02	50·5	4·40	3·28	59·50
Melt	C.732	22·20	45·6	4·79	1·69	12·25
Melt	C.735	14·64	49·4	5·60	2·46	30·28
Melt	C.736	9·28	53·2	4·45	3·87	85·40
Melt	C.739	14·20	51·5	5·02	2·68	38·50
Tap	C.739	11·88	48·6	3·86	3·09	47·60
Melt	C.740	17·80	44·2	4·90	1·95	13·65
Melt	C.733	16·00	47·2	6·48	2·10	18·20
Melt	C.741	11·72	47·5	4·79	2·88	44·10
Melt	C.742	18·40	43·0	5·25	1·82	13·83
Melt	C.744	23·00	41·8	5·30	1·48	8·05
Melt	C.745	18·36	47·2	7·49	1·82	13·41
Melt	C.746	15·24	44·2	6·16	2·06	18·03
Melt	C.747	19·44	42·0	4·89	1·73	10·85
Melt	C.748	14·80	46·8	4·79	2·39	24·85
Melt	C.749	18·60	43·4	6·40	1·74	10·85
Tap	C.749	13·60	49·8	4·48	2·76	43·40
Melt	C.750	19·04	44·6	5·48	1·82	12·08
Tap	C.750	12·46	52·6	4·48	3·10	52·15
Melt	C.743	10·84	48·5	5·13	3·04	50·75
Melt	C.751	9·56	47·7	5·17	3·24	58·80
Tap	C.751	8·32	46·0	3·55	3·88	86·10
Melt	C.752	20·80	44·5	1·93	1·96	12·78
Melt	C.753	14·70	47·3	4·90	2·41	28·53
Melt	C.756	27·10	41·9	1·39	1·47	6·86
Tap	C.756	21·40	47·3	1·58	2·06	16·98
Melt No. 2	C.768	20·24	43·8	9·19	1·49	7·08
Melt	C.768	12·72	56·9	8·34	2·70	33·60
Melt	C.769	20·44	40·7	7·41	1·46	5·78
Melt	C.770	20·96	43·2	6·79	1·56	7·54
Melt	C.771	20·64	42·9	6·87	1·56	7·35
Melt	C.772	20·02	48·1	8·99	1·66	10·50
Melt No. 2	C.773	21·36	43·9	8·25	1·48	7·21
Melt	C.773	19·92	46·0	8·65	1·61	8·68
Melt	C.776	22·16	47·7	6·02	1·69	9·64
Tap	C.772	12·32	49·8	6·18	2·69	35·35
Tap	C.774	15·64	53·6	5·40	2·54	36·05
Tap	C.775	16·88	52·8	3·94	2·54	35·00
Tap	C.776	13·28	54·4	3·20	3·30	60·20
Melt	C.777	24·16	45·8	3·86	1·63	9·45
Tap	C.777	14·12	55·2	2·97	3·24	59·50
Melt	C.778	21·32	39·6	4·52	1·53	9·45
Melt	C.779	18·52	45·0	5·06	1·91	15·05
Melt	C.780	22·80	41·2	3·05	1·59	8·93
Melt	C.781	23·44	43·9	5·10	1·54	7·70
Tap	C.781	15·44	52·9	4·05	2·72	35·35
Melt	C.783	30·72	43·3	1·62	1·34	5·81
Tap	C.783	16·96	54·7	3·17	2·72	37·10
Melt	C.784	31·04	43·3	1·81	1·32	5·95
Melt	C.785	17·04	43·2	6·50	1·82	14·00
Tap	C.785	12·36	49·7	5·17	2·84	37·80
Melt	C.788	22·80	40·9	4·63	1·49	6·86
Tap	C.788	13·44	48·3	4·55	2·68	33·60
Melt	C.759	16·96	48·5	6·75	2·05	18·90
Melt	C.760	16·56	44·4	7·33	1·86	14·53
Melt	C.761	16·12	47·9	4·52	2·32	24·68
Melt	C.763	16·52	47·4	5·94	2·11	20·48
Premelt	C.764	19·24	44·8	8·87	1·65	9·63
Melt	C.765	19·44	44·2	6·64	1·70	11·02
Melt	C.766	20·12	40·6	6·56	1·52	7·56
Melt	C.767	24·72	40·6	5·17	1·36	6·65
Melt	C.789	23·96	41·4	4·87	1·43	6·30
Tap	C.789	12·92	52·8	4·79	2·98	40·25
Melt	C.801	22·80	42·2	4·79	1·53	8·75
Melt	C.802	16·60	42·5	6·02	1·88	12·60
Melt	C.803	19·04	39·4	5·25	1·62	10·15
Melt	C.804	22·04	40·8	4·32	1·55	10·15
Melt	C.805	15·84	44·3	4·90	2·14	19·95
Melt	C.806	19·80	40·7	4·40	1·68	12·25
Melt	C.807	18·56	43·3	5·56	1·80	12·25
Melt	C.808	21·68	46·8	4·75	1·77	10·15
Tap	C.808	12·36	52·6	3·51	3·32	54·60
Melt	C.810	18·48	49·8	4·32	2·18	21·35
Melt	C.811	13·80	44·6	4·48	2·44	29·05
Tap	C.811	11·44	48·0	3·47	3·22	52·50
Melt	C.812	14·32	47·4	4·21	2·56	30·80
Melt	C.813	14·56	46·0	4·36	2·43	26·60
Melt	C.816	15·28	46·5	4·09	2·40	28·00
Premelt	C.819	23·20	35·7	4·09	1·31	6·09
Melt	C.817	16·80	42·8	5·64	1·91	14·18
Melt	C.820	20·60	40·5	4·48	1·62	10·85
Melt	C.822	16·80	47·9	3·90	2·32	25·20
Melt	C.826	16·04	48·3	4·78	2·33	25·20
Melt	C.827	19·88	42·2	5·25	1·58	9·98
Melt	C.829	16·72	43·0	4·78	2·00	14·00
Melt	C.830	19·30	44·6	5·25	1·81	9·80
Melt	C.832	18·92	42·8	3·86	1·88	14·00
Melt	C.833	20·80	40·8	3·71	1·67	11·55
Melt	C.834	16·80	41·6	4·40	1·96	16·80
Tap	C.834	14·70	46·8	3·78	2·54	33·25

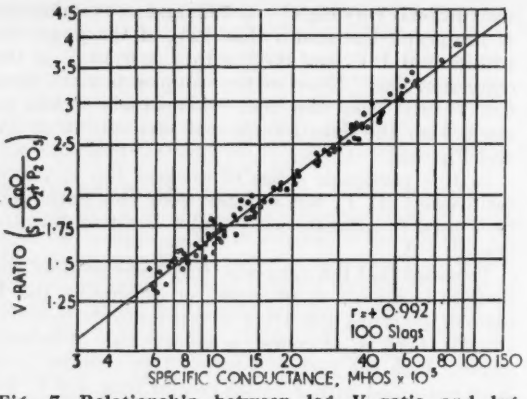


Fig. 7—Relationship between log *V* ratio and log specific conductance of water extracts (0·2 g/100 ml)

V ratio derived from rapid analysis, the method is accurate to $\pm 0{\cdot}15$ *V* in 85% of the cases, as compared with 95% within $\pm 0{\cdot}15$ *V* under the very strict control exercised in collecting the data for Fig. 7.

No attempt is made to explain why it should be possible to estimate the slag *V* ratio so closely by the conductance method nor why the relationship should obey a law of the type $V = aK^b$, where $V = V$ ratio, $K =$ specific conductance, and a and b are constants.

Both from theoretical considerations and from analyses of some water extracts, it is clear that the slag constituent primarily responsible for the conductance effect is CaO, and it appears that the extent to which the CaO is leached out of compounds and solid solutions in the solid slag is in some way proportional to the *V* ratio.

SUMMARY

From the investigations into the *p*H method for estimating slag basicity the conclusions which have been drawn are :

(1) Reproducible results can be obtained using a normal glass electrode standardized with a 10·55 *p*H buffer solution, but the electrode is quickly attacked by the basic solution ; an alkali-resistant " Alki " electrode standardized with a 9·23 *p*H buffer is to be preferred.

(2) The *p*H measurement is affected by the particle size of the slag powder, and the most consistent results are likely to be obtained when the sample is ground to pass completely a − 120-mesh screen.

(3) The *p*H measurement is affected by the weight of slag per given volume of water up to about 0·5 g per 100 ml of water, after which increasing sample weight has no effect on the *p*H reading.

(4) With 0·5 g per 100 ml of water, the *p*H value is constant after 10 min shaking time, thereby confirming the findings of Philbrook.

(5) With the technique developed from this work, an examination of a series of 73 slags showed that the *V* ratio can be estimated to within $\pm 0{\cdot}3$ *V* in only 75% of the samples, and 1 sample in 20 is more than 0·5 *V* ratio in error.

It has been concluded that the method is not sufficiently accurate to provide a reliable check on pancake estimates of *V* ratio or to be of much practical use for slag control.

Experiments have shown there there is a close relationship between the logarithm of the slag *V* ratio and the logarithm of the specific conductance of a water extract of the ground slag. The factors of importance affecting the conductivity measurement are :

(1) It is recommended that the slag-powder sample be ground to pass completely a − 120-mesh screen.

(2) The specific conductance of the extract rises sharply with increasing weight of slag, and a longer shaking time is needed for constant conductivity at higher concentrations. A weight of 0·20 g of slag per 100 ml of water has been adopted in the final procedure.

(3) Time of shaking before measuring the conductivity affects the measurement up to 15–25 min, according to the sample weight. With 0·20 g per 100 ml of water, the reading is constant after 15 min shaking.

(4) The slag suspension must be filtered before making the measurement on the extract and, to eliminate the effect of acid retained in the paper during manufacture, the filter paper should be thoroughly washed with CO_2-free distilled water before filtering the suspensions.

(5) The specific conductance of the extract changes approximately 2% per ° C, and the temperature of the extract should be adjusted to 18° C before measuring the conductivity.

(6) The use of specially prepared conductivity water is not necessary, but it is preferable to use CO_2-free water, freshly prepared, and kept in a stoppered flask.

The *V* ratio of the slag may be estimated by the conductance method within $\pm 0{\cdot}15$ *V* in 85%–95% of the samples, provided the slag is homogeneous and free from undissolved lime. If solid particles of CaO are present in the sample the estimation may be in error by as much as 0·9 *V*. The estimation takes one worker 20–23 min from receipt of the sample in the laboratory. The technique is very simple and may be put into operation with the minimum training of personnel, but close attention to the details of procedure are essential for the best results.

No great length of experience of the application of the method to slag control has yet been gained, but the indications are that it can provide a speedy and accurate check on visual methods of estimation.

Acknowledgments

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A Note on the Varying-Turbulence Cowper Stove

THE DENAIN-ANZIN TESTS AND THE C.S.I. STANDARD COWPER STOVE

By Daniel Petit

SYNOPSIS

There are two main principles of a varying-turbulence Cowper stove : (1) The turbulence must be feeble in the upper part so as to accumulate calories over a great height, and very great in the lower part so as to retain the calories in the stove by a sort of filter. (2) The openings must be large enough to avoid stoppage and to allow easy cleaning.

Detailed tests carried out at Denain on a stove of this type are reported. The results obtained show the usual efficiency test to be insignificant and have led to a new and simple method of determining the optimum operating conditions with precision and certainty.

Details are given of a standard Cowper stove specially designed for use with large-capacity blast-furnaces.

THE construction of Cowper stoves has made little progress in the last twenty or twenty-five years, although many "zoned" stoves, in which the cross-section of the openings decreased from top to bottom of the checkerwork, were built in Europe before the war. It has been shown that the principles on which these stoves were based were open to much doubt, and in practice it was often found that the small openings became stopped up, owing to the inevitable fall into the checkers of pieces of bricks or mortar.

In the United States of America the conditions under which the works operate are quite different, and simple Cowper stoves which generally would not be suited to European needs have been used.

For a long time attempts were made to build uniform-turbulence Cowper stoves, especially in Germany, but because of the obvious disadvantages, which were confirmed in practice, they have been generally abandoned.

In order to achieve high efficiency and a large accumulation of heat in a small volume, it is necessary to build a stove in which the turbulence is feeble in the upper part so as to accumulate calories over a great height, whilst it is very great in the lower part so as to retain the calories in the stove by a sort of filter. Further, the openings must be large enough to avoid the risk of stoppage and to allow easy cleaning. These are the main principles of the varying-turbulence Cowper stove.

Very detailed tests have been made on a 108-ft high, 23-ft 11-in dia stove of this type at the Denain Works of the Société des Forges et Aciéries de Denain-Anzin. More than 150 million kilo-calories (595,000,000 B.Th.U) can be accumulated in it, that is, two such Cowper stoves serving a furnace would in effect constitute a "gas-holder" with a capacity of more than 10,594,000 cu.ft. of gas at 112 B.Th.U./cu.ft.

This large heat capacity has made it possible at Denain to remain "on blast" for periods of up to 11½ hr with a temperature drop of only 210° C on the air leaving the stove, this temperature being regulated by addition of cold air before it reaches the blast-furnace.

In practice, in most of the projects now being made in France, the Cowper stoves are arranged to have a heat capacity sufficient to enable the gas supply to be completely cut off for 8 hr/day with two stoves serving each furnace.

The graphs obtained at Denain have shown how insignificant are the efficiency tests usually made. They have led to a new and very simple method of calculation for Cowper stoves, such that the optimum conditions of operation for a given installation can now be determined with precision and certainty.

Starting from the Denain results, the design for a standard Cowper stove, specially designed for use with large-capacity blast-furnaces, has been established. Its construction is simple and cheap ; with a coke consumption of 590 tons/day and a blast temperature of 700° C the gas supply can be cut off completely from the stove for 8 hr/day, at the same time maintaining excellent efficiency.

PRINCIPLES OF THE VARYING-TURBULENCE COWPER STOVE

Theorists and practical men are in general agreement in condemning zoned Cowper stoves. As early as 1935 Monsieur Derclaye wrote : "The two-zone stove offers no special advantage over the single-zone stove. If the Cowper stove is to be improved, the problem must be attacked along different lines." Even at that time Monsieur Derclaye suggested that turbulence should be introduced to improve the Cowper stove.

Constant-turbulence stoves have long been known. They have been abandoned because, even more so than the ordinary single-zone stove, they led to an accumulation of heat in the head of the stove which was very prejudicial to the results obtained. What was needed was, so to speak, to distribute the calories over a fairly high zone and to make in the lower part a sort of "filter" that would prevent the escape of calories. This result had to be achieved with bricks

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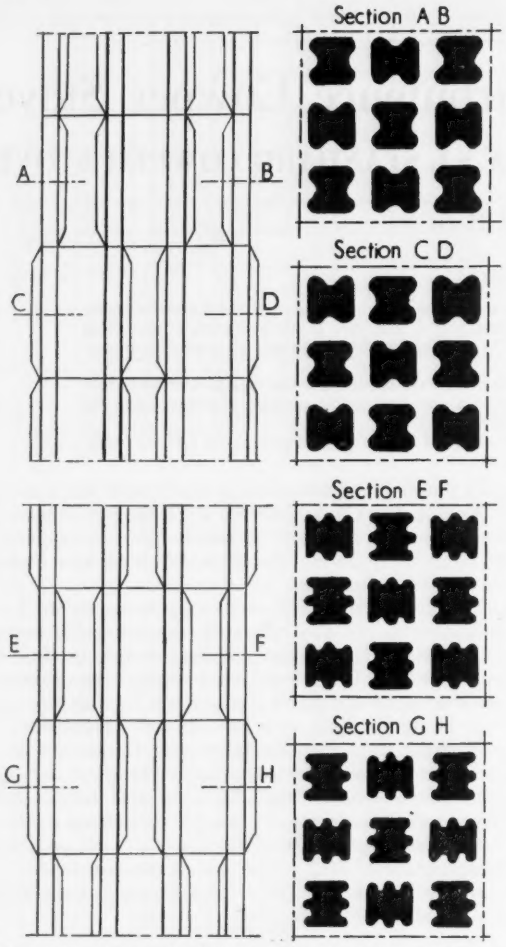


Fig. 1—Checkerwork of the varying-turbulence Cowper stove

of simple shape and with free openings from top to bottom such that they could be easily cleaned.

A Cowper stove is intended to last for twenty or twenty-five years, and to remain efficient for the whole of the time it is in use.

In the zoned stove, in the course of time, the small openings in the lower part become stopped up with chips off the bricks or mortar debris, as a result of which either the efficiency is reduced or all the checkers must be taken out and cleaned.

The checkerwork in the varying-turbulence stove has square openings with rounded corners. Orifices of this shape have most generally been adopted and have given the best results. Round holes lead to bad utilization of refractory material, and with hexagonal holes the blocks had projecting fins that were difficult to make and were easily broken.

The square openings of the varying-turbulence stove have two or four ridges (*see* Fig. 1). It is by changing the relative positions of the blocks with these openings that the turbulence is caused to vary.

In a stove such as that at Denain (*see* Fig. 2) the checkerwork is made up as follows :

At the top, 11 ft 7 in of blocks, $1\frac{1}{2}$ in thick, which serve particularly to protect the upper part of the checkerwork ; 13 ft 1 in of blocks with two ridges, placed all in the same direction ; 16 ft 4 in of blocks with two ridges, reversed in pairs ; 19 ft 8 in of blocks with two ridges, reversed in each course ; and 24 ft 7 in of blocks with four ridges, reversed in each course, in order to obtain maximum turbulence.

The lower part constitutes the " filter " and so, as it were, retains the calories in the stove.

The varying-turbulence stove is usually constructed with orifices inscribed in a square of $2\frac{3}{4}$-in side, and with a minimum thickness of 1 in. If the gas used is very clean, orifices inscribed in a $2\frac{3}{8}$-in square may be used. These $2\frac{3}{8}$- or $2\frac{3}{4}$-in orifices are made in blocks with five holes, which, as will be seen later, lead to great stability.

To simplify the construction, blocks with a single hole, with orifices inscribed in a $3\frac{1}{2}$- or 4-in square and with a greater thickness, can be used. Table I shows the characteristics of these various checkers.

A well-built stove should meet the following requirements :

(1) Its checkerwork should be self-supporting, that is, it should not lean against the outer wall of the stove.

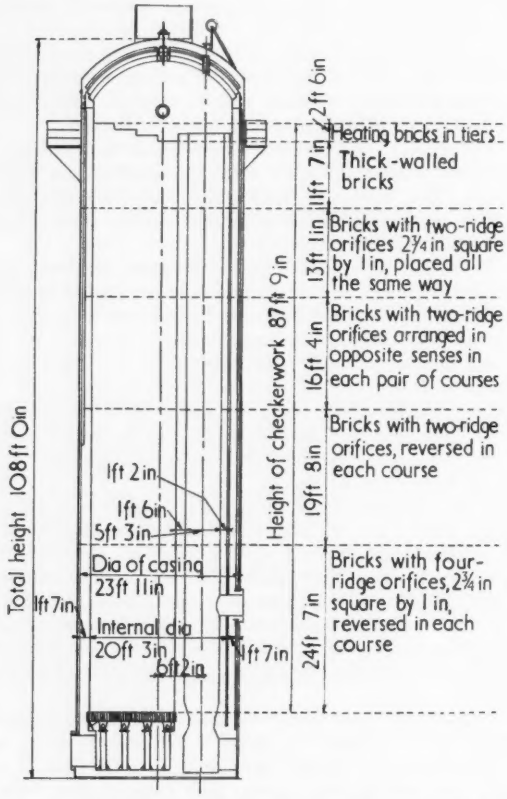

Total height 108 ft 0 in
Height of checkerwork 87 ft 9 in
2 ft 6 in
11 ft 7 in
13 ft 1 in
16 ft 4 in
19 ft 8 in
24 ft 7 in
Heating bricks in tiers
Thick-walled bricks
Bricks with two-ridge orifices 2¾ in square by 1 in, placed all the same way
Bricks with two-ridge orifices arranged in opposite senses in each pair of courses
Bricks with two-ridge orifices, reversed in each course
Bricks with four-ridge orifices, 2¾ in square by 1 in, reversed in each course
1 ft 2 in
1 ft 6 in
5 ft 3 in
Dia of casing 23 ft 11 in
1 ft 7 in
Internal dia 20 ft 3 in
6 ft 2 in
1 ft 7 in

Fig. 2—Varying-turbulence stove. Arrangement of checkerwork at Denain

(2) It should be built up of independent columns.
(3) It should have continuous vertical orifices, without intercommunicating channels, and be easily cleaned.

It is essential that the lining and the checkerwork should be able to expand independently, particularly in the upper part. Lack of appreciation of this principle has led to serious troubles, notably in the tubular stove. Blocks with five holes will meet this condition perfectly.

The irregularity of the shape of the bricks causes them to bear more or less on each other, so that horizontal displacement of any column is not possible. Many tests have been made to prove this. Horizontal bonds or bricks not arranged in columns should be avoided in stove construction, (i) because the inevitable irregularities in the height of the blocks cause the elements of the checkerwork to bear more or less unevenly, and stability is lacking ; and (ii), above all, because this design is dangerous for checkerwork of aluminous materials which undergo after-contraction in use.

This system of construction is particularly to be avoided when the stove has horizontal communicating channels which set up irregular heating of the upper zone. The result is irregular after-contraction, which is the cause of the distortion that has been observed on stoves, particularly in Germany.

In checkerwork built up in independent columns after-contraction cannot cause trouble in service simply because the columns are independent. This consideration applies with particular force to stoves that are to be heated to high temperatures or used as heat accumulators. To obtain regular distribution of the heat in the checkerwork the orifices must be continuous and without intercommunications.

The waste gases in effect leave the combustion chamber at such a rate that, as is well known, they tend to sweep towards the side of the stove opposite to the combustion chamber. In checkers with horizontal channels the upper part at least is heated irregularly and the checkerwork is not used efficiently. Vertical channels, on the other hand, cause a "chimney effect" which distributes the gases almost perfectly. The efficiency of this "self-regulation" in the circulation of the air and waste gases has been shown by experience at Denain.

TESTS AT DENAIN

The Société des Forges et Aciéries de Denain-Anzin was the first in France to adopt the varying-turbulence Cowper stove. The principle was applied initially to two stoves on which construction had been started in 1943. Owing to circumstances it was not possible to put the first one into operation until early 1947.

Requirements of the Stoves

Two stoves were to supply a blast-furnace capable of making 492 tons of pig iron per 24 hr, with a coke consumption of 492 tons ; the temperature of the cold blast was 40° C, that of the hot blast was to be able to reach an average of 750° C.

The stove was to heat 116,500 cu.ft. of air per ton of coke consumed. The dust content varied from 0·57 to 0·85 mg/cu.ft.

Table I

VARYING-TURBULENCE CHECKERWORK

Weight of checkerwork per cubic foot (sp.gr. 1·85), 63·6 lb ; "porosity," 45%

Orifice Inscribed in:	Min Thickness, in	Heating Surface per cu.ft. of Checkerwork	
		With two ridges, sq.ft.	With four ridges, sq.ft.
4-in square	1½	7·306	8·212
2¾-in square*	1	9·429	10·335
2⅜-in square†	1	10·647	12·176

* Checkerwork with "2¾-in square openings" is usually employed
† This last checker size is used only if the casings available are too small and the gas is very clean

The new stoves were to be built in the existing casings, 108 ft high and 23 ft 11 in dia.

In view of the working conditions in the Denain works, relatively short of gas, the new stoves were to have the largest heat capacity possible.

Arrangement of Checkerwork

The general arrangement of the checkerwork adopted has been stated above (*see* Fig. 2). Its height was 85 ft 3 in ; in the upper part the checkers were built in three circular steps or tiers, the axes of which coincided with the axis of the combustion flue, in order to facilitate the distribution of the waste gases. Other details were as follows :

Useful horizontal surface of the checkerwork	264 sq.ft.
Total volume of checkerwork	22,500 cu.ft.
Total weight	Approx. 640 tons
Total surface of the checkerwork*	Approx. 223,000 sq.ft.

* As a result of turbulence the whole of this surface could be heated efficiently.

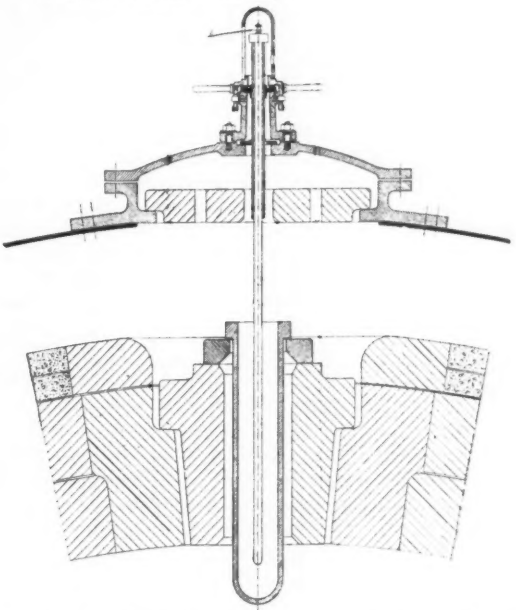


Fig. 3—Varying-turbulence stove. Arrangement for measuring the temperature of the dome

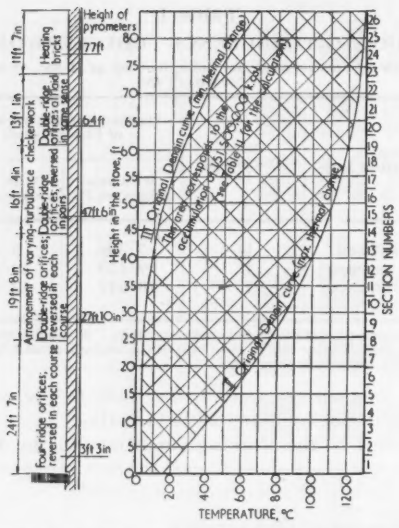


Fig. 4—Distribution of heat in a varying-turbulence stove

The Denain stoves have no automatic control on the valves. They are fitted with a pyrometer in the dome, as shown in the diagram, and with other usual metering or control devices, including a blast-temperature regulator.

Arrangement of Thermocouples

The first Cowper stove at Denain was fitted with numerous thermocouples by means of which the temperature of the checkerwork could be determined under different working conditions to enable the distribution of the calories to be studied. The arrangement of the thermocouples is shown in Fig. 4. They were placed at heights of $3\frac{1}{4}$ ft, $27\frac{3}{4}$ ft, $47\frac{1}{2}$ ft, 64 ft, and 77 ft above the grid. At each level there were three thermocouples, one at the centre of the stove, one on the diameter passing through the combustion chamber but at the opposite end from it, $3\frac{1}{4}$ ft away from the outer wall, and the third on the diameter at right-angles, also $3\frac{1}{4}$ ft from the wall. Each of the couples corresponding to these points was arranged in a small cavity placed vertically at the point of intersection of four walls. Despite all precautions, many of the couples were destroyed in a short time, but as the necessary readings were taken quickly the desired results were obtained.

SUMMARY OF RESULTS

Vertical Temperature Distribution

The tests were started on 29th April, 1947. The stove was first of all "charged" thermally to the maximum, this "charge" corresponding to: (*a*) A temperature of 1280° C in the dome and at the top of the checkerwork, and (*b*) a waste-gas temperature of about 200° C.

Under these conditions the readings of the thermocouples produced curve I in Fig. 4. Heat was then extracted from the stove until the blast temperature reached 500° C. Curve III of Fig. 4 was then obtained. The area between these two curves represents the calorific charge of the stove; knowing the specific heat of the bricks at the temperature considered, it can be easily calculated. This calculation is summarized in Table II; it gives 151,579,426 kg.cal., *i.e.*, approximately 150 million (595,000,000 B.Th.U.).

During these tests the remarkable result was achieved of a period on blast of $11\frac{1}{2}$ hr (*see* Fig. 5) with a temperature drop of only 210° C, the volume of blast blown during that period being 1,270,000 cu.ft./hr.

Dome Temperature

The dome thermocouple was situated as shown in Fig. 3. It was placed in a carborundum sheath in such a way that it projected 2 in below the bottom surface of the arch.

During the whole of the period on blast there was a constant difference of about 200° C between the temperature of the air measured as it left the stove and that registered by the dome pyrometer (*see* Fig. 6). It was also found that the dome temperature was much the same as that of the upper part of the checkerwork.

Finally, when the stove was put back on gas the temperature at first increased very rapidly (100° C during the first $\frac{1}{4}$ hr), but afterwards it rose more and more slowly, as is usual.

Horizontal Temperature Distribution

To check the horizontal distribution of temperature the thermocouples were placed as already described. The difference of temperature between three couples situated at the same level was usually nil, or at the most about 10–20° C.

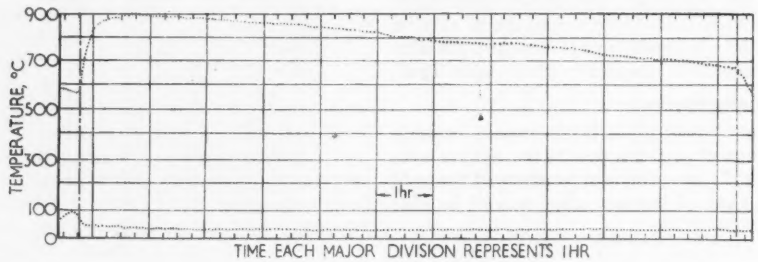


Fig. 5—Denain Cowper stove. Reversal lasting $11\frac{1}{2}$ hr on 30th April, 1947

Table II

CALCULATION OF THE THERMAL CAPACITY OF THE DENAIN COWPER STOVES

N = number of the section 3½ ft high (*see* Fig. 4)
T = average temperature of the sections
C = specific heat corresponding to this temperature
P = weight of active refractory in one section

N	Curve III (Fig. 4)			Curve I (Fig. 4)		
	T	C	$T \times C$	T'	C'	$T' \times C'$
1	40	0·195	7·80	200	0·210	42·00
2	40	,,	7·80	300	0·220	66·00
3	40	,,	7·80	370	,,	81·40
4	40	,,	7·80	450	0·230	103·50
5	40	,,	7·80	540	0·240	129·60
6	40	,,	7·80	610	0·250	152·50
7	40	,,	7·80	690	,,	172·50
8	40	,,	7·80	760	0·265	201·40
9	50	,,	9·75	820	0·280	229·60
10	50	,,	9·75	880	,,	246·40
11	60	,,	11·70	930	0·300	279·00
12	70	,,	13·65	970	,,	291·00
13	90	,,	17·55	1000	,,	300·00
14	110	0·200	22·00	1050	,,	315·00
15	140	,,	28·00	1090	,,	327·00
16	160	,,	32·00	1110	,,	333·00
17	200	0·210	42·00	1150	,,	345·00
18	230	,,	48·30	1170	,,	351·00
19	260	,,	54·60	1190	,,	357·00
20	300	0·220	66·00	1220	,,	366·00
21	350	,,	77·00	1240	,,	372·00
22	400	0·230	92·00	1250	,,	375·00
23	450	,,	103·50	1260	,,	378·00
24	510	0·240	122·40	1270	,,	381·00
25	580	,,	139·20	1280	,,	384·00
26	660	0·250	165·00	1290	,,	387·00
			1116·80			6965·90

P = 24·5 sq.m. × 1020 = 24,990 kg.
Add corresponding active part of lining and combustion chamber 925 kg.
25,915 kg.

$P \times T' \times C'$ = 180,521,298 kg.cal.
$P \times T \times C$ = 28,941,872 ,,

Difference = 151,579,426 ,,
(approx. 595,000,000 B.Th.U.)

Waste-Gas Temperature

When the stove is lightly "charged" the temperature of the waste gases is naturally low—of the order of 65° C. Above a certain charge it rises, of course, until it reaches about 200° C when the maximum thermal charge is attained.

Other Tests

When the first tests had been finished a general survey was made covering the whole of the month of May, 1947, the stove being used always as a heat reservoir. Figure 6 relates to the working on 7th May, which may be regarded as a sort of average.

For the whole of May the mean temperature of the waste gas was 90·6° C. The excess air being practically nil, this corresponds to a loss in the waste gas of about 5¼%.

The means available at the Denain works did not allow of the accurate determination of the losses through the steel casing nor, therefore, of the efficiency, but on the basis of results obtained in other tests it can be estimated that in the Denain stoves, which are well insulated, the losses by radiation should be not more than 3 or 4%. The efficiency during May thus varied between 90 and 91%.

It may be added that the burner usually consumed 706,000 cu.ft., occasionally 883,000 cu.ft., of gas per hour, with a calorific value of 112 B.Th.U./cu.ft.

CONCLUSIONS

The following analysis of the Denain test results is made in order that certain practical conclusions may be drawn from them.

Useful Heat Reservoir

The considerable heat reservoir of the varying-turbulence Cowper stove is explained by the very principle of the appliance and is reflected in the shape of the curves obtained, particularly by curve I (Fig. 4). Measurements of the temperature at various heights in the checkerwork that had so far been used in other stoves gave diagrams of similar shape to curve III (Fig. 4), so that the thermal charge in the appliance was low (*see* Fig 7). The advantage of the varying

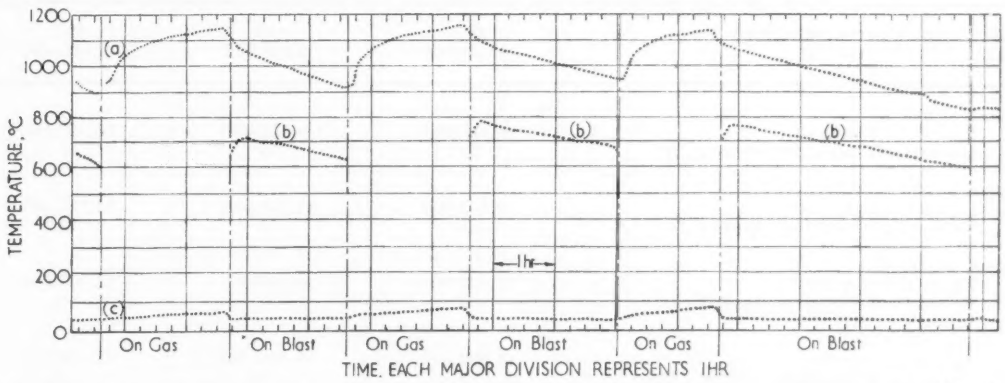


Fig. 6—Denain Cowper stove. Blast chart for normal operation, 7th May, 1947. (*a*) Temperature of the dome. (*b*) Blast temperature on leaving stove. (*c*) Temperature of waste gases. Note the low waste-gas temperature and the parallelism between the blast and dome temperatures

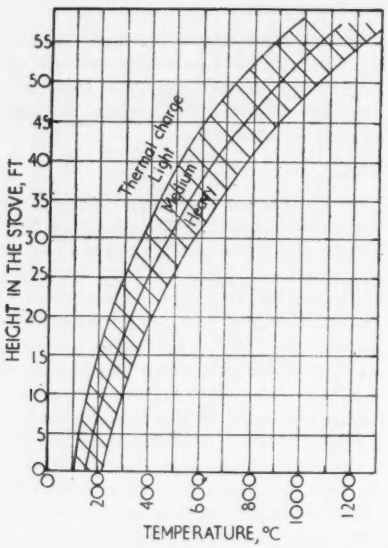


Fig. 7—Heat capacity of a Cowper stove (according to German documents)

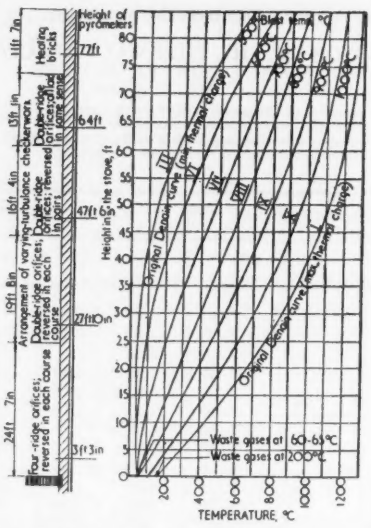

Height of pyrometers
Heating bricks
Arrangement of varying-turbulence checkerwork
Double-ridge orifices, reversed in each course
Four-ridge orifices; reversed in each course
Height in the stove, ft
TEMPERATURE, °C
Original Cowper curve (no thermal charge)
Waste gases at 60-65°C
Waste gases at 200°C

Fig. 8—Distribution of heat in a varying-turbulence stove. Heat capacity at different blast temperatures at Denain

turbulence is shown by curve I, which is concave instead of convex. This large charge makes it possible to envisage long periods on blast.

Figure 8 and Table III summarize the results that can be obtained with the Denain stoves working under different conditions.

From the author's many studies made so far on the other stoves it appears that even with existing casings of normal dimensions it should be possible in most works to cut the gas entirely from two stoves in operation for a period of 8 hr—a matter of great value in the majority of plants.

Efficiency

One of the important conclusions to be drawn from the Denain tests is that the efficiency determinations usually made on Cowper stoves have only an extremely limited significance. Figure 9 shows curves recorded at Denain (curves I and III). The hatched area along these two curves represents the number of calories necessary to operate on blast for 1 hr. It is at once apparent that if an efficiency test with blast at 500° C were carried out along curve III the results would be altogether different from those obtained in another test run along curve I, that is, on a charged stove. In the first case the waste-gas temperature would be about 65° C and the waste-gas loss about 4%, whilst in the second case the corresponding figures would be about 200° C and 11%. For the same stove there could be a difference in the efficiency of 7%. The

Table III
CALCULATION OF THE MAXIMUM DURATION OF THE ON-BLAST PERIOD*

Furnace Operation:	Slow	Normal	Fastest
Coke consumption, tons/24 hr	330	443	492
Blast volume, cu.ft./hr	1,625,000	2,185,000	2,430,000
Blast at 500° C			
Kg.cal./hr required	6,716,000	9,033,000	10,037,000
Theoretical duration, approx., hr	22	16	14
Blast at 600° C			
Kg.cal./hr required	9,372,000	11,261,000	12,512,000
Theoretical duration, approx., hr	14	11	10
Blast at 700° C			
Kg.cal./hr required	10,028,000	13,488,000	14,987,000
Theoretical duration, approx., hr	10	7	6
Blast at 800° C			
Kg.cal./hr required	11,500,000	15,469,000	17,188,000
Theoretical duration, approx., hr	6	4¾	4½

* These figures take no account of heat losses which are partially compensated for by smaller losses in the dome due to the reduction in the number of reversals

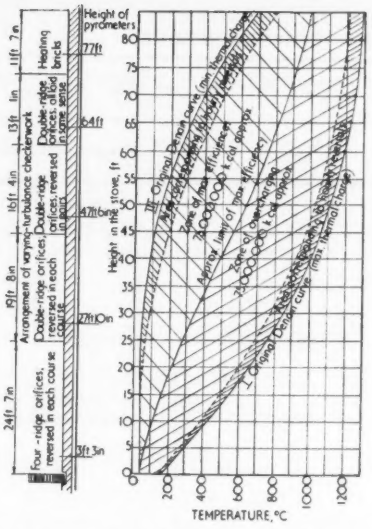


Fig. 9—Distribution of heat in a varying-turbulence stove at Denain. Calculation of heat capacity

efficiency of any given stove would appear to be good or bad simply according to whether the test was made when the thermal charge was light or heavy. The line in the middle of Fig. 9 represents roughly the limit of what might be called the maximum efficiency. This diagram gives valuable indications concerning the mode of operation to be chosen for any given works. If the gas consumption of all the services is regular it would be desirable to use lightly charged stoves working at maximum efficiency, but if, on the contrary, the works consumed much gas during, say, the morning shift, the stove would be worked as an "accumulator," even if the efficiency were not so good.

Quoting from Denain by way of example, Table III gives the maximum theoretical duration for the on-blast period of a Cowper stove. In practice, lower values would be adopted.

CONSTRUCTION OF "STANDARD" COWPER STOVES

To equip the large-capacity blast-furnaces under construction in France, the author has studied a

Table IV

"STANDARD" STOVE. CALORIES AVAILABLE AT GIVEN BLAST TEMPERATURES

Blast Temp, °C	Kilo-Calories available at:	
	Max Temp 1280° C	Max Temp 1200° C
500	195,000,000	162,500,000
600	162,500,000	130,000,000
700	130,000,000	97,500,000
800	97,500,000	65,000,000
900	65,000,000	32,500,000
1000	32,500,000	..

Table V

THEORETICAL DURATION OF ON-BLAST PERIOD*

Blast Temp, °C	Approx. Theoretical Duration of On-Blast Period, hr, for:	
	Max Temp 1280° C	Max Temp 1200° C
Blast-Furnace Coke Consumption 492 tons/day		
500	195,000,000 / 10,175,000 = 19¼	162,500,000 / 10,175,000 = 16
600	162,500,000 / 12,512,500 = 13	130,000,000 / 12,512,500 = 10¼
700	130,000,000 / 14,918,750 = 8¾	97,500,000 / 14,918,750 = 6½
800	97,500,000 / 17,325,000 = 5½	65,000,000 / 17,325,000 = 3¾
Blast-Furnace Coke Consumption 590 tons/day		
500	195,000,000 / 12,210,000 = 16	162,500,000 / 12,210,000 = 13¼
600	162,500,000 / 15,015,000 = 10¾	130,000,000 / 15,015,000 = 8¾
700	130,000,000 / 17,902,500 = 7¼	97,500,000 / 17,902,500 = 5½
800	97,500,000 / 20,790,000 = 4¾	65,000,000 / 20,790,000 = 3
Blast-Furnace Coke Consumption 688 tons/day		
500	195,000,000 / 14,245,000 = 13¾	162,500,000 / 14,245,000 = 11½
600	162,500,000 / 17,517,500 = 9¼	130,000,000 / 17,517,500 = 7½
700	130,000,000 / 20,886,250 = 6¼	97,500,000 / 20,886,250 = 4¾
800	97,500,000 / 24,255,000 = 4	65,000,000 / 24,255,000 = 2¾

* These figures take no account of heat losses through the casing

"standard" Cowper stove 26¼ ft in dia and 108¼ ft high. The principles of its construction are obviously applicable to appliances of smaller capacity.

Space does not permit of a full description of this new stove, but the following are its principal features :

Horizontal surface of checkerwork	340 sq.ft.
Height of checkerwork	85¼ ft
Volume of checkerwork	29,000 cu.ft.
Weight of checkerwork	824 tons
Surface : With 2¾-in orifice	287,000 sq.ft.
With 2⅞-in orifice	331,000 sq.ft.

Table VI
REFRACTORIES REQUIRED FOR A C.S.I. COWPER STOVE

	Weight of Refractories, tons				Volume of Insulation, cu.ft.	
	20–25% Al_2O_3	25–30% Al_2O_3	30–35% Al_2O_3	35–40% Al_2O_3	Density = 0·6	Density = 0·4
Dome			74·2		243	265
Lining, upper part	138·8		132·9		1300	
Lining, lower part	236·0					
Lining, below the grid	50·3					
Combustion chamber			188·0			
Combustion chamber, below the grid	17·1					
Totals	442·3		395·1		1543	265
Checkerwork (85 ft 3 in.):						
11 ft 7 in thick-walled blocks				110	Insulating grit and powder 3248 cu.ft.	
13 ft 1 in bricks with 2 ridges, laid same way			126·2			
16 ft 4 in bricks with 2 ridges, reversed in pairs		158·4				
19 ft 8 in bricks with 2 ridges, reversed in each course	191					
24 ft 7 in bricks with 4 ridges, reversed in each course	237					
Totals	428	158·4	126·2	110		

Total weight of refractories = 1660 tons

The vertical constitution of the checkerwork is exactly the same as that at Denain, so that its calculation was considerably simplified. The data are summarized in Tables IV and V.

The construction of a "standard" Cowper stove would need a whole paper to itself; the following, however, are its main characteristics:

(*a*) The cast-iron grid is of very simple design (patented); all the parts are interlocked, so that they cannot come apart.

The supporting columns are also easily manufactured.

For a stove 26¼ ft in dia the weight of cast iron is less than 39½ tons.

(*b*) The lining can be made either of simple shaped blocks or of rectangular bricks.

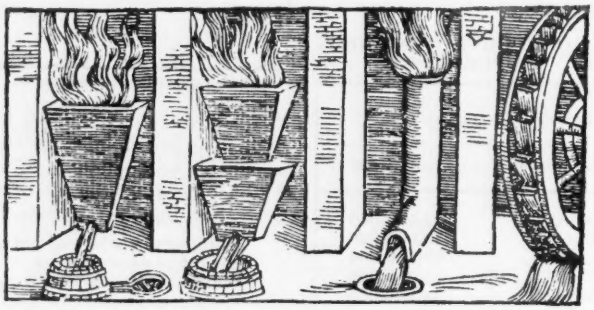
In the lower part up to the level of the grid the bricks are bonded with special high-strength cement.

(*c*) The combustion chamber is completely independent; it is built of a small number of shaped bricks. Its weight is evenly distributed and it is constructed of refractories that do not suffer after-contraction, so that there is no danger that it will warp towards the interior.

(*d*) The insulation of the lining and the dome is specially reinforced.

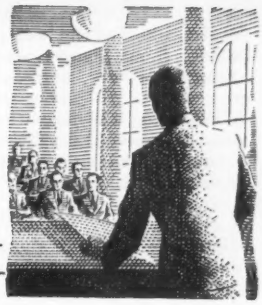
(*e*) For the most part the checkerwork is constructed of machine-made bricks, so that their cost is very low and the output of the works making them is extremely high.

Table VI shows the quantities of refractories of various qualities required for the construction of a C.S.I. Cowper stove.



Various forms of blast-furnaces (reproduced from the *Pirotechnia* of Vannoccio Biringuccio)

DISCUSSION ON PAPERS



DISCUSSION AT MEETINGS · WRITTEN CONTRIBUTIONS

Joint Discussion on the Papers—

A COMPARISON OF MOULDS OF STANDARD COMPOSITION AND OF APPROXIMATELY INGOT MOULD SUB-COMMITTEE COMPOSITION* *by W. L. Kerlie*

MOULD-WEIGHT/INGOT-WEIGHT RATIO AND ITS RELATION TO MOULD CONSUMPTION† *by N. H. Bacon*

INGOT SURFACE DEFECTS‡ *by P. Walker*

Dr. **L. Reeve** (Appleby-Frodingham Steel Company): It gives me great pleasure to open the discussion on these three papers. The two ingot-mould papers illustrate the growth of another new technique in the steel industry. Dr. Desch has already commented on the development of new laboratory techniques, and I should like to refer to the growth of statistical methods for the examination of works data. These two ingot-mould papers are an interesting example of this trend, and Mr. Bacon, in particular, who has been very interested in these methods, must be pleased to see the growing tendency to use them throughout the steel industry. Statistical methods are well known in other sciences and in other industries, and I am sure that their application to the steel industry will lead to valuable conclusions. It is sometimes said that statistics will prove anything or nothing, but by a judicious use of statistical methods in conjunction with works experimental methods, that is, controlled experiments under works conditions, a good deal of very valuable information will, I am certain, become available and will be of great benefit to the iron and steel industry.

Mr. Kerlie's paper is an illustration of a statistical problem of a common type, where two groups of data are available, representing two methods of making a particular item, in this case ingot moulds.

He was aiming at two different analyses, but, in the way in which these things do happen in actual practice, there is an actual variation in the analyses in both groups, so much so that some of them overlap; there is also a variation in the lives of the moulds concerned, the averages differing by a relatively small amount, *viz.*, eight lives. It becomes a problem in simple statistics to determine whether this difference is significant; that is, whether it has arisen by chance or whether it represents a definite trend. The author has shown that it actually means something, in spite of the relatively small differences in the lives of the two sets of moulds, and he goes on to determine the specific influence of silicon and phosphorus on mould life.

Nevertheless, the so-called Ingot Mould Sub-Committee analysis should not, in my view, be taken as invariably the best and as applying to all sizes of moulds. It was largely worked out, I believe, for ingot moulds of about 4 tons' capacity; in our experience with moulds of 10 and 15 tons' capacity some change towards slightly higher silicon values gives rather better results.

Mr. Bacon's paper is an even more complete exercise in statistical methods. There were three major variables to deal with: the mould weight itself, the mould/ingot ratio, and the mould consumption. By the method of partial correlation the author shows clearly that for constant mould/ingot ratio the effect of mould weight on mould consumption is relatively unimportant—non-significant as the statistician puts it. The important variable is the mould/ingot ratio. Our own experience, using mould/ingot ratios of 0·8 to 1·0 for moulds of 6 tons and upwards, broadly supports his conclusions.

I think that the use of the data shown in Fig. 3 of Mr. Bacon's paper is open to criticism. The diagram shows mould weight plotted against the mould/ingot ratio and he has rather arbitrarily separated data below 130 cwt. in weight and re-plotted it on a larger scale in Fig. 5. The obvious separation, judging by the two groups of figures in Fig. 3, should be appreciably higher, at about 190 cwt. Such a division may not give such a good correlation, but I think that in fairness it should have been done. The upper group of figures at 300 cwt. and upwards is obviously a separate group, and a logical separation of such data is justified on statistical grounds, but an arbitrary one, at 130 cwt., is open to criticism.

Finally, it would help those who are new to statistical methods if definitions of the terms n, r, and p, were given in the paper, or alternatively a reference made to a suitable textbook.

Mr. Walker's paper is, I think, a model short paper, condensing the maximum of useful information into the smallest possible space. More of such papers would be welcomed at these meetings, where unfortunately the reverse is more often the case. I also welcome it because it is a contribution from that very hard-worked person about whom we hear too little, the works metal-

* *Journal of The Iron and Steel Institute*, 1947, vol. 157, Nov., p. 410.

† *Journal of The Iron and Steel Institute*, 1948, vol. 158, Jan., p. 81.

‡ *Journal of The Iron and Steel Institute*, 1948, vol. 158, Jan., p. 96.

lurgist, who has the headache of trying to find out why certain things happen under works conditions and not under laboratory conditions. It is also a welcome sign of a renewed interest in the old subject of ingot defects, following on the formation by the British Iron and Steel Research Association of the Ingot Defects Sub-Committee, which is heir to all the troubles of previous Committees which dealt with the same subject.

The "box" formation, to which the author refers, is not unknown to us; we have had some ingots which were quite as bad as those which he shows. Obviously they should not occur in a well-organized works, but they do. However, by attention to teeming practice, and in the case of larger ingots by the use of splash cans, they can be largely, though not entirely, stopped. In our experience, these "boxes" lead finally to "shelly" types of defect rather than to "seamy" defects in the final product. In our experience shelly defects, in both plates and sections, tend to concentrate in the bottom half of the ingot, and that is supported by the fact that a splash would also tend to be confined to the bottom half.

Mr. Walker very briefly refers to the technique which he used in obtaining the photomicrographs of what he calls the "corner section." I should like to indicate the method for cutting such sections introduced by the Ingot Defects Sub-Committee, and which is now being used by various members for the study of these ingot defects. It is simply a convenient method of attacking the problem of sub-surface defects. Figure A is a diagram showing the method of gas-cutting specimens from the corner of the ingot, roughly 6–8 in. in length; Fig. B is a photograph of a 10-ton ingot with the ingot corner cut out; and Fig. C is a transverse cross-section of the ingot corner, showing sub-surface blow-hole formation in a rather thick-skinned "balanced" type of ingot.

The technique has, of course, some points of disadvantage compared with the more normal method of completely sectioning an ingot, but it is quick and cheap and it provides a great deal of information about sub-surface conditions. It has been shown that the corner gives a reasonable indication of these conditions in the rest of the ingot skin at the same height. Finally, and this is valuable, the method allows the ingot concerned to be rolled subsequently and any defects in the final product can then be correlated with the information obtained from the corner section. In that connection, I should like to ask Mr. Walker, with regard to the particular ingot which he illustrates, showing the double skin, whether he actually followed it through to the final product and whether an actual shelly or seamy defect was isolated.

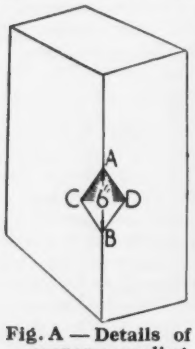


Fig. A — Details of corner-sampling method

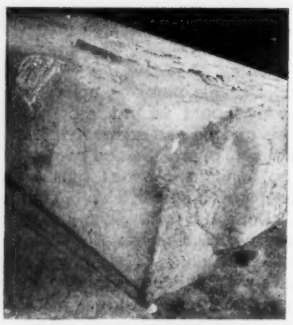


Fig. B—Ingot corner sampling. General view of 10-ton ingot

Fig. C—Ingot corner cross-section. One-half actual size

Mr. **W. H. Glaisher** (British Iron and Steel Research Association): Dr. Reeve has already brought up one of the points which I was going to make, which is that the results which Mr. Kerlie has obtained would not have been obtained simply by looking at his figures. They would not have been obtained completely even by drawing scatter diagrams, as the author has done in Figs. 3 and 4. He has obtained additional information by the use of statistical methods.

One extra piece of information which the methods used did not give, but which can be obtained from the data, is that the effects of phosphorus and silicon may not be the same with different compositions; *e.g.*, the effect of phosphorus with high silicon content may be different from its effect with low silicon. In order to see whether there was any information on this point in the data, I split up all the moulds of both compositions into two ranges of silicon content: those below 1·5% and those of 1·5% and above; I then divided each of those groups into phosphorus contents, obtaining the following figures:

	Life with <1·5% Si	Life with >1·5% Si
Low phosphorus, *i.e.*, <0·1%	107(20)	96(8)
High phosphorus, *i.e.*, >0·1%	121(18)	98(12)

(Figures in brackets relate to the number of moulds)

This information is not obtained from the analysis of variance. I am not suggesting that those figures alone prove that the effect of phosphorus with low silicon is different from that with high silicon, but they do bear out other results which have been submitted to the Ingot Moulds Sub-Committee in two instances where effects of a similar sort were found. The effect of the lower silicon content is greatest with high phosphorus content, and the effect of high phosphorus is most marked with low silicon. There is very little effect with silicon above 1·5%, but it is quite considerable with silicon below 1·5%. I think that this is an important point, and emphasize it because the same effect has now been found in three instances with moulds of this type.

Why do silicon and phosphorus have these effects on moulds? Unfortunately, we are not in a position to say. It may be because of some effect on the microstructure or macrostructure, or on the mechanical properties, but so far attempts to correlate mould life with any of these have not given results of much value. It is one of the matters on which further progress is very desirable.

I should like to mention one point in connection with Mr. Bacon's paper: the effect of mould-weight/ingot-weight ratio on the ingot itself. For any given ingot weight this ratio is determined largely by the thickness of the mould wall. This subject has not been discussed in any detail at this meeting. I have been through a great deal of the literature on the subject to try to find some concrete evidence to show whether thicker walls, for instance, produce more rapid solidification, or not, but

I have not been able to find much evidence from experimental work. The best information that I could find was in a paper by L. H. Nelson to the American Society for Metals in 1934. Nelson did some work on the tipping of 20-in. ingots, the ingots being inverted at various times after casting, and he found that with different mould-wall thicknesses there was only a small effect on the thickness of ingot wall; *e.g.*, with a mould-wall thickness increased by about 40%, from 4·6 to 6·4 in., there was a very slight increase in the amount of metal solidified in 20 min. or more. The moulds used were from 40 to 100% thicker at the bottom than at the top, and although the steel is in contact with the wall at the bottom a minute or two earlier than with the wall at the top, the differences in ingot-wall thicknesses even after more than 20 min. were only of the order of 1–5%.

This is another of the problems to which no complete answer is available, and one on which advances could be made by means of controlled experiments.

Mr. **R. D. Lawrie** (The Stanton Ironworks Co., Ltd.): As I have been associated with Mr. Kerlie in the manufacture of the moulds which are the subject of his paper, I do not wish to enter into the discussion, but rather to hold myself in readiness to answer any questions pertaining to the manufacture, which might arise out of the discussion.

On first reading the paper, I wondered if the moulds had been cast from the same melt, and, given similar analyses, whether the results would have been better for the purpose of comparison; but afterwards, on reflection, I thought that perhaps the variation in analysis such as had been obtained might enable mould users to decide on the analysis which gave the best results, and so be a guide for the future.

With regard to the question raised by Mr. Bacon on variation in thickness of moulds, I can only state that moulds made by Stanton are considerably nearer to drawing size that those of which Mr. Bacon spoke.

Mr. **A. Bevan** (Guest Keen Baldwins Iron and Steel Co., Ltd.): The experiments which Mr. Kerlie has described have since set in motion a good deal of work. At Cardiff we attempted a comparison between moulds containing high silicon and low phosphorus, low silicon and low phosphorus, and low silicon and high phosphorus. Taken as a comparison, the results were inconclusive, but when examined as a whole they showed a big improvement in life as the silicon content fell to 1% and less and as the phosphorus rose to 0·2% and even more. We actually had some moulds with up to 0·5% of phosphorus and some of these had the longest lives of all. The very high phosphorus content certainly did not appear to have a detrimental effect.

Mr. Glaisher has mentioned that the effect of phosphorus increases as the silicon decreases, the converse also being true. This was also noted in our experiments.

I should like to take up Dr. Reeve's point about the Ingot Moulds Sub-Committee's specification not being the "gospel truth." This is an important point as nearly all the experiments that have been carried out have been tried with 4-ton square moulds, bogie cast, and not slab moulds. The indications are that for larger moulds, slab moulds, and moulds subjected to bad pit-casting practice, modifications to the specification will be necessary. This is a field in which a considerable amount of work has yet to be done.

Mr. **W. L. Kerlie** (Messrs. Stewarts and Lloyds, Ltd., Corby): I should like to suggest to Mr. Bacon, for his consideration, that in coming to a conclusion with regard to the desirable mould/ingot ratio we might well consider the mould as being in three parts. For example, if the mould has a semi-closed top we imagine that it is cut off at the top, and the mould/ingot ratio is based on the lower portion. With semi-closed bottom moulds the same principle would apply, and with moulds with a hot top the effective mould/ingot ratio would be based on that length which contains solid steel throughout.

Mr. Bacon shows in Fig. 19 of his paper the optimum mould/ingot ratio for ingot moulds of more than 18 cwt. I submit that with more curvature on the bottom right-hand part of the curve, so as to be more or less asymptotic with regard to the mould/ingot ratio axis, he would get a curve of which an extension would represent the smaller moulds.

During a visit to France I obtained some data on ingot moulds on the smaller range of sizes, and these had lives of up to 600 heats, and after machining the base sometimes 1000 heats, or figures of that order. These results are given in a special paper which has been given a restricted publication by the Ingot Mould Sub-Committee, and they confirm, or tend to confirm, Fig. 19, which I think on extension would cover the whole range of ingot-mould sizes.

CORRESPONDENCE

Mr. **E. Walker** (Bakewell, Derbyshire) wrote: I should like to congratulate Mr. Bacon on his work in dissecting the mould/ingot ratio in relation to mould consumption for the whole of the German industry. His task has been eased somewhat by having access to the records of Ristow.

The Ministry of Supply were wise in sending an investigating team to Germany in 1946 to secure Report B.I.O.S. 685. The team, which was led by Mr. Bacon, consisted of three ingot-mould makers and two ingot-mould users. The combined report, both as to mode of manufacture and use of moulds in Germany, contains a mass of technical detail which will be of vital interest to the steel trade of this country in further reducing in the future the consumption of ingot moulds per ton of steel produced. An omnibus investigation of this character has never been undertaken in this country on so extensive a scale.

As one of the team dealing with the manufacture of moulds, I can endorse that in a general way the internal surface finish of the moulds was particularly rough and not by any means up to the standard required in this country. However, there was one smaller foundry—Gebruder Westphal, of Peine—producing 200 tons per month, of excellent surface finish both inside and out. These moulds were all consumed locally by Ilseder Hutte, Peine, and it was found from the records of Ristow that these moulds gave the lowest consumption in the whole of Germany per ton of steel produced.

On further investigation at the steelworks of Ilseder Hutte, it was found that the whole of their range of ingot moulds had been designed scientifically, according to a pre-arranged standard. All the details of this are given on p. 96 of B.I.O.S. Report No. 685. It is worth studying and shows the close co-operation between user and producer over a period of years. A similar system to this could be adopted in this country to the advantage of the steel trade.

With the results so far achieved by the Ingot Mould Sub-Committee, together with B.I.O.S. Report No. 685 for comparison, and the results of this valuable paper, the Ingot Mould Sub-Committee should now be in a position to examine the whole range of drawings of ingot moulds in this country and to suggest amendments to design to obtain the optimum mould/ingot ratio.

As a maker of moulds I would say that there are in existence too many types for the same size in ingots, which does not help mass production. As in Germany, so in this country, each works designs its own moulds and the maker is in a very difficult position to criticize.

Now seems to be the time for the closest co-operation between the users and makers on the subject of design, the best analysis, and the quantity and quality of hematite iron used in the mixture.

I feel certain that if all these variables could be standardized a considerable reduction in the consumption of ingot moulds throughout the steel industry could be achieved in a short space of time, as ingot-mould consumption is a considerable item of cost in steel production. This would then satisfy conclusions (2) and (3) drawn by Mr. Bacon in his paper.

Mr. Bacon has given us, maker and user alike, an important and additional contribution to the science of ingot moulds. It is vital that these results should be quickly applied to the industry.

The layout of the first part of B.I.O.S. Report No. 685, dealing with the manufacture of ingot moulds in the various foundries visited, has been done in rather a novel way for ease of comparison, and as far as I am aware this method has never been attempted before in these reports. It has much to commend it in giving quick comparison of methods in each works.

All the investigations were determined to produce a full and comprehensive report which would bear investigation and criticism from the technical experts of this country. However, paper restrictions and the methods of publication have not done justice to our efforts. But this has been somewhat overcome by the mass of documents and drawings which were brought back by the team and which are listed in the Appendix (p. 96) and the Addenda (pp. 123 and 124); these are available for inspection.

The full benefit of this report will not be apparent for some years to come until new foundries have been erected.

At some date in the future the Ministry of Supply is bound to authorize the erection of a new foundry or foundries for the large-scale production of ingot moulds and steelworks castings. The usual method is to give the job of design of buildings and layout of plant to consulting engineers. Any defects and omissions are not apparent until afterwards and are costly to rectify. I suggest the Ministry makes the design and layout a co-operative effort of consulting engineers, ironfounders, foundry experts, and members of the team to give advice; also, if necessary, one or two German experts who have had practical experience in the design of European foundries during the last ten years. Only then, by a combined effort of this description, will the steel trade in this country have the finest mass-production foundries in the western hemisphere.

I trust that these few words of mine, as an engineer and a practical founder, will be taken heed of at the highest level of decision. I shall then consider that the effort the team put into their report was worth while for the benefit of all concerned.

AUTHORS' REPLIES

Mr. **W. L. Kerlie** replied: The correspondence from Mr. E. Walker raises some controversial issues. I would submit that the close co-operation he mentions has existed in this country for a considerable number of years. Some exception might be taken to the suggestion made in the final paragraph. I should think that the foundry experts in this country are fully alive to the needs of the iron and steel industry and are perfectly capable of designing foundries for the large-scale production of ingot moulds.

Mr. **N. H. Bacon** replied: Dr. Reeve's criticism regarding Fig. 3 would appear, on the face of it, to be justified. Owing to the wide range of mould weights (33–425 cwt.) the small scale used in Fig. 3 has resulted in the moulds of weight less than 230 cwt. being bunched together very

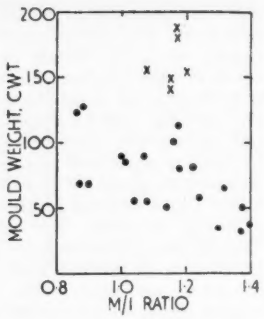


Fig. D—Lower part of Fig. 3 of Bacon's paper plotted on a more open scale

closely. This bunching has somewhat obscured the abnormality of the mould/ingot ratios for moulds of weight 130–187 cwt. If the points are plotted on a wider scale this abnormality becomes clearly apparent (*see* Fig. D).

It was certainly an oversight that no explanation of the terms n, r, and p was given; they are as follows:

n = number of pairs of observations.

r = correlation coefficient; where $r = 1$ the correlation is perfect, *i.e.*, the dependent variable (*e.g.*, in Fig. 3, the mould/ingot ratio) would vary directly with the operating variable (mould weight) for every observation. With a complete absence of correlation or co-variation r would be zero.

p = probability, *i.e.*, the odds against the r value being due to pure chance, *e.g.*, in Fig. 2, where $r = 0{\cdot}45$, $p = 0{\cdot}01$, there is only 1 chance in 100 that the r value obtained is simply due to a fortuitous grouping of the points.

Mr. Glaisher has given some very interesting information as to the effect of wall thickness on ingot structure. There appears to be no doubt that many moulds have been and are made with walls up to 70% thicker than is necessary, with the idea that wall thickness has some practical effect on the speed of solidification of the ingot. From all the evidence so far available the effect, for all practical purposes, is negligible.

Mr. Lawrie's claim that moulds made at Stanton are close to drawing size as regards wall thickness is of great importance, since if the present variations found in certain makes, *i.e.*, up to 1-in. and sometimes up to as much as 2-in. difference between one wall and another in the same mould, could be eliminated, then the mould/ingot ratios could, with safety, be still further reduced.

In reply to Mr. Kerlie, I agree that the desirable mould/ingot ratio should be determined with particular reference to the chill or body portion of the mould; superimposed heads and closed bottoms should be treated as a separate problem.

The paper on French moulds, to which Mr. Kerlie refers, confirms, as he says, the conclusions given as to the desirable mould/ingot ratio as set out in Fig. 19, *i.e.*, as follows:

French Moulds (*Mr. Kerlie's paper*)

Capacity, cwt	Mould Consumption, lb/ton	Mould/Ingot Ratio	Mould/Ingot Ratio Given in Fig. 19
57	19·3	0·99	1·03
47	23·3	1·02	1·10
35	37·6	1·38	1·30
16·8	24·7	1·43	1·60
5·6	34·0	2·40	Not given
5·4	25·5	4·88	,,
2·6	50·6	5·39	,,
1·6	40·1	9·11	,,

Mr. Walker comments on the low mould consumption at the Ilseder Hutte plant. The mould/ingot ratios used

were the lowest in Germany, but were still rather higher than appears to be desirable, judging from the data as a whole.

Mr. **P. Walker** replied: I thank Dr. Reeve for his contribution. It is an oversight that I did not mention the part which he has played in developing the corner sampling technique. We have found it a very useful method. With regard to following through an ingot on which curtaining is present, we have not done this, but it is something which we hope to do in the future.

I have received a communication relative to this paper from Mr. T. J. Woods, of the Republic Steel Corporation, Cleveland, Ohio, and a copy of his paper, "*The Origin, Definition, and Prevention of Scabs.*"

Mr. Woods, who is concerned with a 54 × 24-in. slab ingot, considers "curtaining" to be of great significance to rollers of flat products. His method of overcoming the defect is by very fast teeming, compared with what is considered a normal teeming rate in this country.

Joint Discussion on the Papers—

ABNORMAL CREEP IN CARBON STEELS* *by J. Glen*

THE CREEP PROPERTIES OF MOLYBDENUM, CHROMIUM–MOLYBDENUM, AND MOLYBDENUM–VANADIUM STEELS† *by J. Glen*

Mr. **Glen** briefly introduced the two papers.

Mr. **W. E. Bardgett** (United Steel Companies, Ltd.): The author is to be congratulated on his very extensive and systematic investigation, which shows the effect on the creep resistance of a number of factors under certain stress and temperature conditions, the effects of these factors having been previously dealt with in somewhat limited aspects.

I have already contributed‡ at some length, in writing, to the discussion of the first paper, and I should like to confine my remarks, now, largely to the second. There are, however, one or two points in the first paper to which I would again like to refer.

Results were submitted of tests which we had carried out on a low-silicon and a high-silicon steel. These showed the dependence of the order of these steels with respect to creep resistance on the stress and temperature adopted for the test; with a change in the rate of creep used as the basis of comparison, or in the temperature, the order of the two steels with respect to creep could be reversed. In a series of short-time tests, results indicated that phosphorous content had quite an appreciable effect on the creep resistance, increase in phosphorus being distinctly beneficial. These two points are of some importance in connection with the further remarks which I wish to make.

In the first paper, in Fig. 11, p. 510, the author shows the effect of aluminium additions on the creep strength of basic open-hearth casts of mild steel measured in a short-period test. We have carried out a similar type of test on basic open-hearth steels having various aluminium additions, and as a matter of interest I should like to show the results. Fig. A shows these results, together with the results obtained by the author, the points on the broken curve being the results at 48 hours obtained from Table IV and plotted in a similar manner to the results at 5 days, and given by the author in Fig. 11. There is a definite displacement to higher aluminium additions of the transition from low to high creep rate when comparing our curve with that of the author. This would seem to indicate that higher aluminium additions are necessary in the case of our steels to reach the transition from low to high creep rate, but I would not attach any real significance to this since the author's curve is defined by only six points, none of which would fall appreciably outside a band representing the limits of our results. I would say that the two sets of figures agree fairly well.

An interesting point is that, on the basis of these results, coarse-grained steels, marked (C) in Fig. A, are invariably high creep resisting; fine-grained steels marked (F) in Fig. A, are generally, but not always, of low creep resistance. If you have a coarse-grained steel, as pointed out by the author, you can be fairly sure that it will not give abnormal creep, but a fine-grained steel may or may not do so.

Turning now to the second paper, I shall, for reasons of time, have to confine my attention to a few only of the points which I would like to discuss. On p. 39, the author says: "The relationships shown in Fig. 4 may vary significantly when the tests are carried out at higher or lower temperatures." That is a very important statement. No one will disagree with it, but it is one of those things that we are apt to forget when making a comparison between different steels and studying creep results in general.

On p. 41 the author says: "By employing temperatures of 550° and 600° C. and stresses of 4, 6, and 9 tons/sq. in. a clear indication of the effect of these variables was obtained." One can well appreciate the difficulty with which the author was faced in deciding on the stresses and temperatures to adopt. Obviously some indication of the effect of the variables had to be obtained in a reasonable time; but in the absence of a range of stresses and temperatures, which would permit extrapolations or deductions to working stresses and temperatures, the results must be accepted as applying only to the particular test conditions, and generalized statements are to be avoided. It would have been especially interesting if the author had adopted working stresses and temperatures in his examination of the effect of tempering; one is there concerned with the creep behaviour after having put the steel into a particular condition.

On p. 40 the author states that furnace-cooling is detrimental to creep resistance, and in Fig. 25 he shows curves for 0·5% molybdenum and chromium–molybdenum steels in the normalized and furnace-cooled conditions, the steels in the furnace-cooled condition being very much inferior in creep strength. I should

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* *Journal of The Iron and Steel Institute*, 1947, vol. 155, April, pp. 501–512.

† *Journal of The Iron and Steel Institute*, 1948, vol. 158, Jan., pp. 37–80.

‡ *Journal of The Iron and Steel Institute*, 1947, vol. 157, Dec., pp. 579–584.

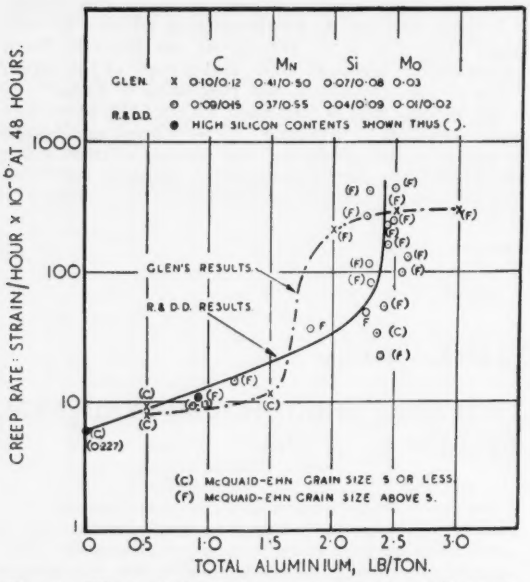
C Mn Si Mo
GLEN. X 0·10/0·12 0·41/0·50 0·07/0·08 0·03
R.&D.D. ○ 0·09/0·15 0·37/0·55 0·04/0·09 0·01/0·02
● HIGH SILICON CONTENTS SHOWN THUS ( ).
CREEP RATE: STRAIN/HOUR x 10⁻⁶ AT 48 HOURS.
1000
100
10
1
GLEN'S RESULTS.
R.& D.D. RESULTS.
(C) McQUAID-EHN GRAIN SIZE 5 OR LESS.
(F) McQUAID-EHN GRAIN SIZE ABOVE 5.
0 0·5 1·0 1·5 2·0 2·5 3·0
TOTAL ALUMINIUM, LB/TON.

Fig. A—Effect of aluminium on creep strength; basic open-hearth steels. Temperature of test 450° C. (842° F.) ; stress, 8·0 tons/sq. in.

like to give some results which we obtained on a low-carbon–chromium–molybdenum steel in the annealed and normalized conditions. The stress in tons/sq. in. for a creep rate of 1×10^{-7} strain/hour, measured at 1000 hours, being :

	510° C.	538° C.
Normalized ...	5·2	4·0
Annealed ...	6·2	4·6

If deterioration on tempering were to be considered, these figures, which showed the annealed steel to have a higher creep resistance than the normalized material, would be somewhat modified. The results, however, at these lower temperatures and stresses show up very differently from those given by the author in Fig. 25. His test conditions were a temperature of 600° C., and a stress of 4 tons/sq. in., the rates of creep in his case being very much higher than the rate to which I have referred.

The author refers to the necessity for very strict control of steel variables, otherwise very wide scatter will be obtained in the results. It may be noted that the open-hearth steels in his Table I show appreciably higher phosphorus, and slightly higher manganese, than the electric steels, and are therefore not strictly comparable according to the data presented by the author showing the effect of manganese, and according to results which we have obtained on the effect of phosphorus, if this effect applies to the 0·5% molybdenum steel.

While on the question of open-hearth and electric steels, it is noted that the author presents data and makes several references to the superiority of open-hearth steels. I would submit that the open-hearth steels may or may not be superior to electric steels, depending on the conditions. The high-frequency steel B5 shown in Fig. 19 (*a*), and compared with the corresponding open-hearth steel B1, was made in a small experimental-type furnace of 18 lb., and it contained lower phosphorus, whilst the test was carried out at a stress and temperature above working stresses and temperatures, giving a high rate of creep. For these reasons, the results do not necessarily give a measure of the relative creep resistance of commercial electric and open-hearth steels at working stresses and temperatures.

The author then states that a 5-ton electric-arc steel, though better, was still inferior to open-hearth steel. In the absence of data, and in view of the many factors which influence creep resistance, this statement has no clear significance. Further, reference is made on p. 50 to the comparison between 18-lb. electric-furnace steel No. 5 and, in this case, open-hearth steel No. 2. Here again we have similar differences in composition; high creep rates are involved, and a laboratory-type experimental cast is compared with a commercial cast. The deductions, therefore, relate only to the particular test conditions. It is not clear why the author should attribute the poor properties of Bailey's steel to its possible origin in an electric furnace, when no information is given of the heat-treatment, composition, and, I assume, method of deoxidation.

The author again refers to the inferiority of electric steels in Fig. 31, on tempering. I should like to give some results which we have obtained on ½% molybdenum steels which have a bearing on these points.

The analyses of the steels were as follows :

	Electric (High-Frequency)	Open-Hearth
Carbon, % ...	0·06	0·18
Manganese, % ...	0·43	0·44
Silicon, % ...	0·270	0·215
Sulphur, % ...	0·021	0·030
Phosphorus, % ...	0·012	0·030
Molybdenum, %	0·57	0·49

The results of the long-time creep tests were :

Temperature		Stress for creep rate of 1×10^{-7} strain/hr at 1000 hr, tons/sq.in.	
° C.	° F.	Electric (H.F.)	Open-Hearth
450	842	11·5	8·2
510	950	6·4	3·1

These results are not intended as a comparison between electric and open-hearth steels, but when carrying out work on electric 0·5% molybdenum steel many years ago, as a matter of interest, we obtained an open-hearth steel of unknown origin, but which the tube manufacturer believed to be representative, and these figures show the creep resistance of the two steels. The electric steel had a creep strength of 6·4 tons/sq. in. at 510° C. for a creep rate of 1×10^{-7} strain/hr., measured at 1000 hours, while the figure for the open-hearth steel was only 3·1 tons/sq. in. These results show that a very strict control must be maintained over those variables which affect creep if the making of steels of this open-hearth type is to be avoided.

Figure B shows the creep curves on four steels, two acid open-hearth and two 10-ton-electric. The carbon content of steel D is only 0·07%, being lower than the others, but the other electric steel is more comparable with the acid open-hearth. The acid open-hearth steels have higher manganese and phosphorus contents, which would be beneficial to creep, and lower silicon and molybdenum contents which would be detrimental. These results show that for the two open-hearth steels the creep rate in the early stages is very much higher than it is for the electric steels, the figures being 16·6 and 20·4 against 11·1 and 3·6 $\times 10^{-7}$ strain/hr. for the creep rate at 24–48 hours, but at 1000 hours the creep-rate figures were 2·0 and 1·1 $\times 10^{-7}$ strain/hr., respectively, for the two acid open-hearth steels A and B, 1·1 $\times 10^{-7}$ strain/hr. for the electric steel C, and 1·0 $\times 10^{-7}$ strain/hr. for the electric steel D, with the lower carbon.

Figure C shows the results of creep tests carried out on

an electric 0·5% molybdenum steel at 510° C., and stress of 3 tons/sq. in., curve A being for the steel in the normalized condition, and curve B for the same steel normalized and reheated 167 hours at 650° C. In spite of the long time of heating at 650° C., which I think we should reckon as equivalent to at least 100,000 hours at a working temperature of 510° C., the steel has shown no deterioration in creep resistance. The test on the sample which was reheated at 650° C., was carried on for over 2000 hours just to see whether it would start to increase in creep rate, but it went on continuously decreasing. The sample which had been reheated at 650° C. did show a very slightly higher creep rate than the sample normalized only.

On the basis of these results, there is no indication that open-hearth steel is superior to electric steel at working stresses and temperatures, and any deterioration evident at high testing temperatures and stresses may be insignificant at working stresses and temperatures.

Figures 28 (*a*), 29 (*a*), and 30 in the paper, dealing with the effect of silicon, show that, as the test temperature is decreased towards the service temperature, the higher silicon steel, which was inferior at the higher temperatures to the lower silicon steel, becomes superior.

On p. 52 the author states that aluminium improved the creep resistance of molybdenum steel in a short-time test. I think that according to Fig. 26 this relates only to molybdenum contents of up to 0·25%.

Figure 32 indicates that if tests were carried out at lower stresses, to give deformations of the order of those permissible in practice, the optimum silicon content would increase.

The author rightly draws attention to the importance of the subject of deterioration. The principle of applying it to short-time tests is, I think, worthy of the closest consideration, particularly in view of the efforts which are being made at the present time to develop reasonable short-period tests for specification purposes. I hope that the author will pursue this very important work.

Dr. **C. H. M. Jenkins** (National Physical Laboratory): I feel that we are all very appreciative of the author's work given in these two papers. I propose, however, to confine my few remarks to the paper on carbon steels.

The use of the term 'abnormal' in regard to the creep of steels is, perhaps, unfortunate as it introduces some degree of confusion, implying, perhaps, that a particular steel is unsuitable, which is often far from the fact. The British Standards Institution has in B.S. 1271 of 1945 introduced a proof test for the creep behaviour of carbon steel of boiler-plate quality in which creep is tolerated to 50 millionths/hr at 450° C. under a stress of 8 tons/sq. in. It might be assumed (and this assumption would be incorrect) that steels exhibiting a creep rate below 50 millionths possessed normal creep properties. Materials having a creep rate above this value are certainly abnormal. The use of severe test conditions (*i.e.*, higher in stress and in temperature than service requirements) is forced upon us, as for acceptance purposes it is the only method capable of discriminating between casts in a reasonable time. The steel covered by the above specification is intended for use in water drums, and might exhibit a creep rate of 20 to 50 millionths/hr. This type of steel, under low total deformation at the relatively low temperatures existing in water

Steel.	Manufacture	Analysis. No Aluminium Addition. Cr	Mn	Si	P	Mo	Heat Treatment	Grain Size. McQuaid	Actual	Creep Rate x 10^7 Strain/Hr. 24/48 Hrs.	1000 Hrs.	Total Plastic Strain at 1000 Hrs. Percent.
A	Acid Open Hearth.	0·15	0·56	0·23	0·037	0·50	Normalised 920°C	3 & 4	6-8	16·6	2·0	$0·074_2$
B	Acid Open Hearth.	0·14	0·63	0·13	0·030	0·45	Normalised 910°C.	2	7-8	20·4	1·1	$0·053_4$
C	Electric Arc.	0·14	0·47	0·33	0·023	0·63	Normalised 920°C	3 & 4	7-8	11·1	1·1	$0·053_0$
D	Electric Arc.	0·07	0·46	0·27	0·023	0·54	Normalised 920°C.	3 & 4	7-8	3·6	1·0	$0·042_3$

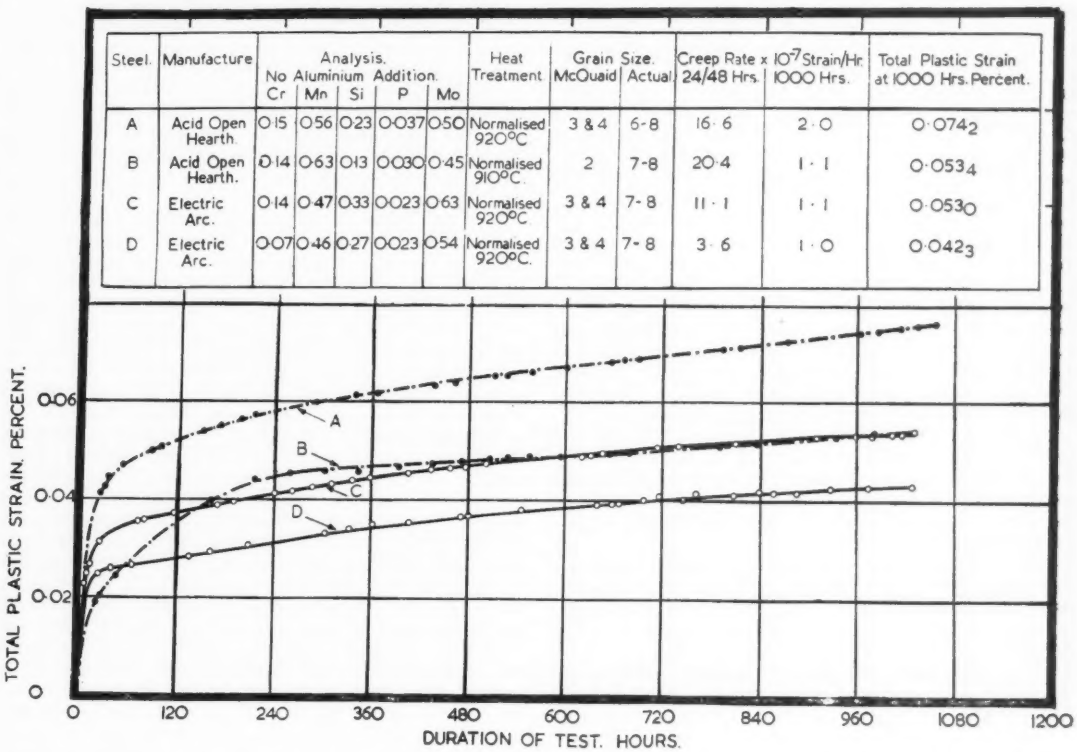

Fig. B—Results of creep tests on four 0·5% molybdenum steels. Temperature of test, 480° C. (896° F.); stress, 9·0 tons/sq. in.

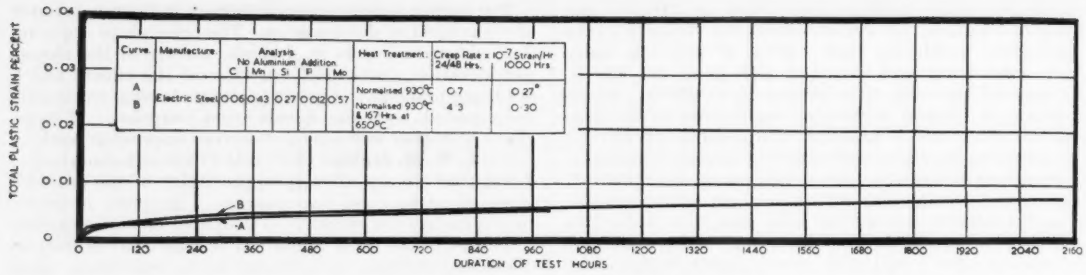
TOTAL PLASTIC STRAIN PERCENT
0·04
0·03
0·02
0·01
0
Curve. Manufacture. Analysis. No Aluminium Addition C Mn Si P Mo Heat Treatment. Creep Rate x 10⁻⁷ Strain/Hr 24/48 Hrs. 1000 Hrs
A Electric Steel 0·06 0·43 0·27 0·012 0·57 Normalised 930°C 0·7 0·27°
B Normalised 930°C & 167 Hrs. at 650°C 4·3 0·30
B
A
0 120 240 360 480 600 720 840 960 1080 1200 1320 1440 1560 1680 1800 1920 2040 2160
DURATION OF TEST HOURS

Fig. C—Results of creep tests on a 0·5% molybdenum steel. Temperature of test, 510° C. (950° F.) ; stress, 3·0 tons/sq. in.

drums, is quite suitable, more so in fact than a steel of more normal characteristics as defined by the standard test conditions. The type of service envisaged is likely to produce creep in the first stage of a creep test, whereas the standard test referred to above compares materials in some portion of the second stage of a creep test.

If a distinction is desirable between normal and abnormal steels, I suggest that it might be placed at between 2 and 10 millionths/hr. under the standard test conditions rather than at the value of 50.

The great difference between the conditions of test of the four carbon steels referred to in Fig. 1 and those of the standard test should be noted. The four steels show 0·1% creep strain in 100,000 hours, whereas the limiting rate of 50 millionths/hr. in B.S. 1271 is 5000 times greater than the average hourly rate of the conditions in Fig. 1 at 450° C. (842° F.).

A further interesting comparison can be made between normal and abnormal steels under conditions which are not improbable in the service life of superheater tubes. In some experimental work carried out, two steels, given a prolonged spheroidizing treatment of 60 days at 650° C., were found to be weaker, an abnormal steel showing a creep rate of 338 millionths per hour under the standard test conditions referred to previously, instead of 120 in the "normalized" condition. On the other hand, a normal steel with a creep rate of 1·0 millionths/hr. was markedly less strong in the heavily spheroidized condition, the creep rate increasing to 237. It is apparent that the normal and abnormal steels had become more or less similar under prolonged spheroidization, and it is deduced that the structural alterations in normal steels proceed slowly, whereas those in abnormal steels proceed swiftly to their final conditions. Indirect evidence of this can be obtained by short-time creep tests at a high stress and a relatively high temperature.

In regard to Fig. 10 the author shows the effect of molybdenum contents of from 0% to 0·14% on the degree of abnormality. It should be pointed out that the creep rates are of a higher order than would be permitted in carbon steels intended for high-temperature service in the region of 400–450° C.

With regard to the preparation of carbon steels for high-temperature service, the addition of aluminium plays an important rôle and the counter-balancing influence of silicon should also be considered. The suggestion is put forward that examination should be made of the oxygen content of the author's steels as this will throw light on their creep behaviour. In some steels which were examined in the National Physical Laboratory it was found that those produced in the B.O.H. furnace had a low oxygen content and a correspondingly high creep rate after deoxidation with a heavy aluminium addition, whereas a high oxygen content was found in steels of low creep rate if the aluminium addition had been light. This relationship was linear in a graph connecting oxygen content and creep rate. No direct relationship could be established in a corresponding series of acid open-hearth steels, but none of these showed the low oxygen contents of basic open-hearth steels heavily killed by aluminium.

Dr. **H. A. Dickie** (Stewarts and Lloyds, Ltd.): It has been known for many years that whilst plain 0·5% molybdenum steel suffers from the unfortunate deficiency of a low elongation at fracture under creep conditions (a kind of hot shortness or hot brittleness), this deficiency has not prevented the successful use of the steel in various applications where the service conditions are presumably well within the capacity of the steel; or it may be in some cases, and perhaps in many cases, where the wastage of the steel by scaling prevents the initiation of surface cracks. Failures, however, have occurred, in several British power stations, in steam pipes which are largely protected from scaling on the outside of the pipes by heat-insulating lagging. These failures have occurred by intercrystalline penetration of cracks through the pipe wall, starting from the outside of the pipes, and have been accompanied by little or no observable extension of the steel.

Various investigators have pointed to a number of unfavourable factors in the particular conditions in question, such as localized stresses of various kinds, built-up stresses, hammer marks, rooting of scale, deficiencies in heat-treatment, or alkaline concentrations on the surface of the pipes arising from the nature and method of application of the lagging. It seems, however, that these factors, or such of them as may have been in active operation, have merely been the straws which broke the back of a somewhat debilitated camel.

Any contribution which can be made to remove, or to minimize the type of hot shortness from which plain molybdenum steel suffers, will be of value to the engineering industry, and the author has made such a contribution by demonstrating clearly the improvement produced by 1% chromium. His results show that the elongation factor is greatly improved and that the life of the steel is considerably lengthened. It would be of value, I think, if the author would consider in future work, determining the minimum quantity of chromium necessary to produce these beneficial changes, since the presence of 1% of chromium (or 0·8%, which is usually employed at present) is not entirely without its objectionable features. The steel in the normalized condition is liable to form martensitic grains instead of pearlitic grains, and at present this can be avoided only by controlling the cooling to a slower rate than normal. It would be best if it were possible to avoid

that, altogether, in conjunction with the beneficial improvement mentioned.

Dr. **R. W. Bailey** (Metropolitan-Vickers Electrical Co., Ltd.): We are very much indebted to the author for these papers, and, I presume, to the President, at whose works the investigations were made. Referring to the author's paper about carbon steels, I consider that we are not making sufficient use of these steels. If abnormality is removed, and if by tests a standard of creep resistance were assured, I am certain there is a good field for the further use of carbon steel. Large quantities of more expensive steels could have been saved in the past if carbon steel had been fully exploited.

With regard to the second paper, a fairly wide range of molybdenum–vanadium steels with differing heat-treatments is being used. As with carbon content, heat-treatment may be chosen to suit the application of the steel. These steels, quenched and tempered, are more effective at lower temperatures, but inferior at the higher temperatures. There are occasions, however, when they are needed at lower temperatures. Considering the limited range of the steels dealt with, the title of the paper is too wide, and it would have been more accurate to have referred to normalized and tempered wrought steels.

Figure 2 (*b*) illustrates this feature very well. This figure gives creep curves for 0·5% molybdenum steels obtained from different sources. Steel No. 5, which is from a casting, is very much below all the other curves, but at about 505° C., a number of the curves bunch together, and No. 5, which has been annealed and given a long furnace treatment after annealing, actually crosses two of the other curves, and it is on its way to run into the other two curves. The point, already made in the discussion, that an annealed steel may give results equalling those for normalized steel, probably refers to a high temperature. I have noticed this characteristic with other steels. Apparently the more " active " a steel is made by heat-treatment or work, the more advantage it shows at the lower temperatures, but it suffers at the higher ones.

The paper contains such a mass of curves, that a sense of perspective is necessary when examining them. What are the essentials ? The curves in which the engineer, and to some extent the steelmaker, are most interested, are those obtained for a specific stress, giving corresponding temperatures and times for a specific creep strain. Extrapolation of these curves gives a good idea of the temperatures at which these stresses would be used in practice. These are the most essential tests. The heating test, where the material is heated to ascertain the extent to which it may weaken, is exploratory and is next in importance. It is essential as a measure of how the material may be expected to behave when it operates at a high temperature for a long time.

With regard to the author's tests which illustrate very effectively the influence of an element such as carbon ; after pointing out the number of variables in the problem, and then taking four or five different steelmaking processes, the author proceeds, on the basis of carbon content, to get a thoroughly consistent set of curves, which is surprising. One may ask why, because as mentioned by the author, the steel suffered a certain amount of plastic deformation upon loading, the lower carbon steel was inferior to the higher carbon steel ?

The process I mentioned for predicting working stresses contains a variable which may easily be overlooked, namely, whereas at the higher temperatures the stress may exceed the proportional limit and produce yielding as lower and lower test temperatures are used, the proportional limit is higher, and a temperature is reached at which there is no plastic strain. It is most probable that considering carbon contents at a high temperature, where there are plastic strain effects, there is one interpretation, and a different one at a lower temperature where plastic strain effects do not operate. We could, by taking carbon contents, associate them with the effects at the different temperatures, but of course, as the author points out, it would mean extensive testing. I do not suggest that the author should make these tests—he has already done very well. Nevertheless the proportional limit is a variable and, generally speaking, working stresses found by the process we have mentioned, and which is referred to by the author, are below the proportional limit at working temperature, and sometimes very much below it.

I have found very low proportional limit values for normalized molybdenum–vanadium steel. This has involved deciding whether to use a working stress which would exceed the proportional limit. This I am prepared to do. If one accepts creep, which is plastic strain, one should not hesitate about accepting a relatively small amount when the stress is applied, although generally there is no plastic strain when stress is applied. In viewing the curves, in which composition is taken as the abscissa (carbon or any other element), one should not forget that the method used to find the working temperature at any particular stress operates at different proportional limits as the temperature is lowered.

Reference has been made in the discussion to a curve for a chromium–molybdenum steel which I had given, and it was suggested that particulars of its manufacture should have been provided. The steel was made by the author's firm, and he may know the process used, which I do not. It was proposed to use the chromium–molybdenum steel for piping at a power station, and my opinion was invited. A representative sample was supplied and the curve referred to showed the result obtained. Our analyses gave a carbon content of 0·11%. The sample was normalized at 875° C., and after heating for different periods at 650° C. gave the results which were reproduced by the author.

There has been some dispute as to whether chromium added to molybdenum steel improves the steel. I think that the author has proved most effectively, from the tests which he has made, that there is an important advantage. It might be suggested that by lowering the carbon content we could take full advantage of the influence of chromium, but, as the author and others know, there have been two cases where, when using low-carbon–molybdenum steel, steam piping has cracked. This experience has made users a little chary of going in for very low carbon contents, and I think wisely so. A carbon content of 0·1% and upwards, yes ; but we must be cautious about going down to the very low carbons which the paper would indicate to be advantageous.

We have had to compare chromium–molybdenum steels with molybdenum steels in the case of bolts, which differ in category from the one embracing the author's steels. In the latter, creep under constant stress takes place as the service continues, but with a bolted joint all the strain is supplied initially as elastic strain, and, as creep takes place, plastic strain replaces the elastic strain with consequent relaxation of load. A high yield-point is therefore an advantage and this favours heat-treatment by quenching and tempering. We made some constant-stress tests on molybdenum-steel bolt material as compared with chromium–molybdenum steel, both steels being quenched and tempered, and found a cross-over of the stress–temperature curves, the chromium–molybdenum steel being superior to the molybdenum steel at the lower ranges of temperature and inferior at

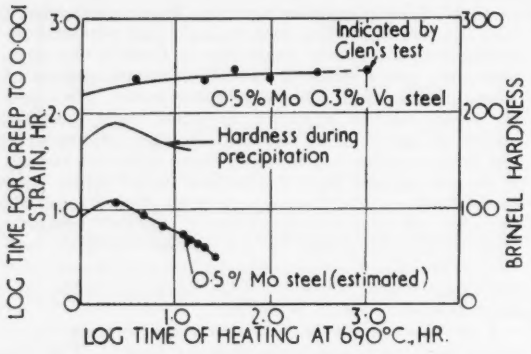

Fig. D—Effect of preliminary heating at 690° C. upon creep. Tests carried out at 4 tons/sq. in. and 600° C.

the higher ranges. A similar cross-over occurs with molybdenum–vanadium steel when quenched and tempered, and when normalized the latter shows up to advantage at the higher temperatures. The results of our tests made before the war gave confidence that the material would more than satisfy the most severe service conditions then contemplated, and we have used molybdenum–vanadium steel forgings for high-temperature steam turbines of the largest powers. This knowledge, in providing a good margin to meet higher operating temperatures, enabled important applications to be made for war purposes. These higher operating steam temperatures are now coming into use. One example in the U.S.A. employs an initial temperature of 1050° F., which is about the maximum operating temperature of molybdenum–vanadium steel for the "first line" life period of 100,000 hours expected from steam-power plant.

The author has spent a great deal of time dealing with rupture tests and has produced some very interesting information. I have always felt that, in the U.S.A., too much weight had been placed on rupture tests, and in the early days of the war, under the exchange of information with America, much of their data was in that form, and it was of but limited usefulness because, for design, creep information relating to small strains was required. If one is to provide for, say, a total deformation of 0·1%, or 0·2%, or 0·3%, one wishes to be assured that fracture will not occur at 0·4%, because, if it may do so, there will be odd places where there will be trouble. But a few tests will ensure that there is a requisite margin, and much rupture testing is unnecessary and a waste of time. Now the point of view in the U.S.A. appears to agree with the one held in this country, and creep-test data are being produced which are similar to those which we have been in the habit of obtaining.

The author has given attention to the influence of size of test-piece, which is very important. I do not think, however, that one can necessarily take advantage of larger size, because there are generally odd corners in the piece where, in spite of the size, a crack may start and, if it does, there are usually conditions which can extend it.

Referring to deriving working stresses, a warning should be given about using the method of estimating the working stresses for, say, 0·1% extension in 100,000 hours, upon the basis of double logarithmic plotting. The author is very cautious about the plotting in connection with the examination of rupture tests, and in this I think that he is right, as there is a risk of being too optimistic. Looking at the curves given in Fig. 69 (*b*) on p. 75, the stresses which are given for molybdenum–vanadium steel appear to me to be definitely optimistic. Engineers have to be cautious in providing for something which may be reached in 100,000 hours, when their tests have not exceeded say 4000 hours. There is always a temptation to extend a straight line, and if one is tempted to do this with a logarithmic plotting the result can be, very easily, much too optimistic. A reasonable interpretation of the data appears to me to fall nearly 20° behind the lower stresses in Fig. 69 (*b*). Our tests of a molybdenum–vanadium steel steam pipe indicate that at 1050° F. (566° C.) a working stress of about 3 tons/sq. in. is reasonable.

The influence of tempering on molybdenum–vanadium steel is very important. The temperature used by the author is one used by me ten years ago in connection with molybdenum–vanadium steel made by the author's firm. It was probably the first sample of commercially produced molybdenum–vanadium steel in this country, and we found that it was distinctly notch-brittle, and engineers do not like notch-brittle material, even if it may not mean anything. Improvement was necessary and we tried tempering, and making both impact and creep tests. After 4 hours' tempering at 690° C. we got a reasonable impact figure and could assure engineers that the material was acceptable and would improve with time; and we explored the effect of other temperatures and times. Attention has been drawn, in the paper, to precipitation-hardening in its bearing upon creep resistance, and in this respect molybdenum–vanadium steel is of interest. This steel displays the usual feature of precipitation-hardening, namely, the hardness increases to a maximum, and with continuation of heating it declines, but remarkably the creep resistance stays up. That is a point of very considerable interest.

Figure D illustrates this point very well, and the author by his very long-time tests is able to give us the extra point for 1000 hours' heating at 690° C., which confirms and extends the conclusion we had reached some ten years earlier, that no weakening by thermal action was indicated. We felt satisfied that no steam plant, in view at the time, would have a combination of temperature and time in service more severe than that represented by the last point at the end of the graph. The author's 1000 hours' heating at 690° C. shows that the top curve does continue without reduction in creep resistance, and that one may depend upon this steel at very high steam temperature for a very long time. The microstructure of the steel undergoes considerable change, but the creep resistance stays up, and in this respect molybdenum–vanadium steel differs very much from the other steels to which the author has referred.

CORRESPONDENCE

Mr. **L. Rotherham** (Royal Aircraft Establishment, Farnborough, Hants.) wrote: In his paper on the creep properties of certain alloy steels Mr. Glen refers on p. 41 to the comparison of short- and long-time creep tests due to Johnson and Tapsell. I think some further consideration of the results of this work should be given since the reasons for the deficiencies in the usual forms of short-time tests will give a guide as to the effectiveness of Mr. Glen's own method of comparing different materials.

Two short-time tests investigated by Johnson and Tapsell gave an order of merit for the different carbon steels the same as for the long-time tests. One of these short-time tests was the time-yield test due to the late Dr. W. H. Hatfield, and the other a German test. The notable feature of both these tests is that, considering the time at which the observation of creep rate is made, the creep rates measured are low, and the stresses not

much higher than normal working stresses. The temperature used for comparing materials may, of course, be the expected working temperature.

I think everyone will agree that creep is a composite process and that it comprises at least viscous grain-boundary flow, quasi-viscous crystalline flow, and normal crystalline slip. All these processes are related to stress, temperature, and time, through quite complicated functions, and it is normally impossible to separate the different contribution to the total deformation. It is for this reason that extrapolation from short to long times is so very dangerous

In the time-yield test we attempted to control the two variables, stress and temperature, since the times involved are so unreasonably long as to be unmanageable. No one will deny that a change from a test of about 10 hours, as in the German standard, to the time-yield test of about 100 hours is an advantage, and further that a test of 1000 hours is still more to be preferred, but only if the other necessary conditions are observed. Mr. Glen has neglected these precautions and, although he has lengthened the time of test, he has increased the temperature and stress, and I feel that his comparisons of material suffer on this account and should not be pressed too far.

Mr. Glen promises more fundamental work on creep in his closing words, and if he makes the comparison of short-time and long-time tests more effective he will have performed a great service.

Turning to the data presented, I think we can all applaud the persistence and thoroughness which has been displayed in this long and difficult study. One thing which leaves me surprised is the very close agreement recorded, in many cases, between the properties for the molybdenum steels and the chromium–molybdenum steels. For instance, if we take the results for the molybdenum steel A2, in Fig. 13 (*a*), and compare them with an equivalent chromium–molybdenum steel B1, in Fig. 19 (*a*), I would suggest that the difference in results is barely significant. If we take the whole mass of data presented, probably Mr. Glen's conclusion that the chromium–molybdenum steels are better than the molybdenum steels is justified.

The difference is not great, however, since if we turn to the rupture-test data on the same two steels at 550° C. and 6 tons/sq. in., which is the nearest test to service conditions, we see that the molybdenum steel survived 5502 hours and the chromium–molybdenum steel broke after 9794 hours. On a logarithmic scale, this amounts to a difference of less than 10%, or on a stress basis to considerably less than 1 ton/sq. in.

When it is remembered that the main defect in molybdenum steel is not its creep resistance, but a lower ductility than carbon steel, I think it would have been of value if Mr. Glen had presented additional data to show that Cr–Mo steel did not suffer in the same way. The very similarity of Mr. Glen's Cr–Mo and Mo steels leads to doubts on the subject. I believe that Cr–Mo steel may have better ductility than molybdenum steel in certain conditions, but that, at the same time, its resistance to deformation may be lower, as suggested by Dr. Bailey. In spite of this it may have a better combination of creep strength and ductility, and thereby have a longer life, and the higher rate of creep may have a positive advantage in the rapid relief of internal stresses.

AUTHOR'S REPLY

Mr. **J. Glen** said in reply : I would like to thank those who have contributed to the discussion for their appreciative remarks, and I might add also that I welcome their criticisms, because in this somewhat complex subject, it is difficult to arrive at conclusions which are valid under all circumstances.

Mr. Bardgett's main criticism refers to the test conditions used to compare the different steels. Admittedly both the stress and temperature used were higher than the working conditions which would be employed, but, after all, 550° C. (1020° F.) is not very much higher than 1000° F., which temperature is now being employed for some of these low-alloy steels. In creep testing it is necessary to decide on the test conditions before any results are obtained, and because of the time involved it is impracticable to repeat all the tests at a wide range of stress and temperature. The test conditions used, however, do show quite clearly that when comparing two steels the one which has the faster creep rate initially may become the better as the test proceeds. It was also shown that the higher the temperature of testing the shorter is the time necessary for this change-over to occur, and *vice versa*. Thus, if the comparison is made at the working temperature, the steel which is the better, even after 10,000 hours of testing, may not necessarily remain the better for a lifetime of 100,000 hours, *i.e.*, at the working temperature it would be necessary to continue the tests for the life period of 100,000 hours to determine which steel was better for service conditions. Since such tests are impracticable, it appears to be necessary, in the present state of our knowledge, to carry out comparative tests at a higher temperature than the highest working temperature which is likely to be employed. I might add, that whereas Mr. Bardgett prefers to carry out his comparative test at the working temperature, Dr. Bailey used a temperature at 600° C. in tests which he described in his discussion. Thus, in choosing a temperature of 550° C. for most of the tests, I have split the difference and can at least say that I have taken the middle course.

Considering Mr. Bardgett's results in more detail, I do not think phosphorus would have such a pronounced effect as he suggests if his results were corrected for the effect of varying manganese, and for this reason it does not seem possible that the effect of phosphorus is sufficient to explain the difference between high-frequency and open-hearth steels. One or two tests which I have carried out support this view, but I would not like to repeat all the tests, again, adding phosphorus as another variable.

I note that Mr. Bardgett's conclusions regarding the relationship between the McQuaid-Ehn grain-size and abnormality agree with my own, and I would also agree that different amounts of aluminium are necessary to provide fine grain-size, depending on the melting practice employed and on the amounts of silicon and manganese added to the steel.

From tests lasting 1000 hours, Mr. Bardgett shows that a furnace-cooled steel is superior to a normalized one, but I consider that these tests would not be conclusive unless they were prolonged for a much longer period as discussed previously. In my opinion the " latent " creep resistance of a steel is lowered the more completely the steel approaches equilibrium conditions, *i.e.*, either by furnace-cooling or by tempering. This is clearly illustrated (curve 5 in Fig. 2 (*b*), of my second paper*) by the inferior results obtained by Tapsell for a 0·5% molybdenum steel casting which had been furnace-cooled and thoroughly tempered.

The comparison shown by Mr. Bardgett in Fig. B between open-hearth and electric-furnace steels shows the electric-furnace steels to be superior in a 1000-hour

* *Journal of The Iron and Steel Institute*, 1948, vol. 158, Jan., p. 38.

test at working stress and temperature. These results do not contradict the data shown in the paper, since it is only after more prolonged testing at these temperatures that the superiority of the open-hearth steel would be detected.

Similarly in Fig. C, showing that 167 hours at 650° C. do not cause deterioration even after 2000 hours, the results agree with the trend suggested in the paper that the lower the temperature of testing the longer the tempering required before deterioration is detected in a short-time test.

With reference to my first paper,* it may be true, as Dr. Jenkins suggests, that the use of the term "abnormality" is unfortunate, since it implies that a particular steel is unsuitable, but in the paper particular emphasis was laid on the definition of abnormality employed. In this definition I seem to be at variance with Dr. Jenkins' own ideas, since he states that a steel showing a creep rate of more than 50×10^{-6} in./in./hr. in a particular creep test (B.S. 1271 : 1945) is certainly abnormal. I agree that it may not be suitable for high-temperature service, but in my definition it is not necessarily abnormal. I consider that abnormality should be defined as the deterioration of creep resistance caused by deoxidation with aluminium and should not be defined in relation to the suitability of the steel for service. Thus, a very low manganese steel may have a creep rate of 100 or more in the above test and yet this result is still normal for this composition even though the steel is not suitable for high-temperature service. Similarly, an abnormal mild steel containing 0·1% molybdenum with a creep rate of about 30 may be suitable for service, but I would still classify it as abnormal, since the same steel without the addition of aluminium would show a creep rate of only 4.

I was very interested in Dr. Jenkins' tests showing that in the thoroughly spheroidized condition there was little to choose between a normal and an abnormal steel, both showing very large creep rates. This I think emphasizes the point that high creep resistance depends on the rate at which a steel approaches equilibrium conditions, and that any treatment which aids this approach to equilibrium is detrimental.

A silicon-killed steel does not usually contain aluminium, but if aluminium is added the steel becomes abnormal in the same way as a low-silicon steel. Thus, silicon of itself has no counterbalancing effect as suggested by Dr. Jenkins.

In answer to Dr. Dickie's query whether less than 0·8% or 1% chromium would be beneficial in minimizing the hot shortness of 0·5% molybdenum steel, I must state that, unfortunately, I have no information on this point, but I consider that even 0·5% would have some effect. His objection to 1% chromium steel is that in the normalized condition there is a tendency to form martensitic grains in open-hearth steels normalized at 950° C. I have not found evidence of this effect, and I would suggest that martensite grains only form when complete diffusion of the chromium carbides has not taken place during heat-treatment. If this is true, then normalizing at a low temperature should tend to exaggerate the effect which he reports, and normalizing at a higher temperature should eliminate it.

Dr. Bailey states that if carbon steels could be guaranteed normal, then greater use could be made of them. I do not see any reason why carbon steels could not be supplied with this guarantee.

When I "predicted" that Dr. Bailey's results on chromium–molybdenum steel referred to a high-frequency steel which had been normalized at a low temperature, I was fairly sure that these tests were carried out on a steel which we had supplied to him many years ago. Although I had an unfair advantage with this prior information, the results do indicate that the effect of composition cannot be stated without also considering the effect of other variables.

Dr. Bailey illustrates other cases of the cross-over effect of stress–temperature curves. These results emphasize once again the necessity of taking care that short-time creep tests do indicate that a steel is suitable for the service required.

With regard to rupture tests, Dr. Bailey states that all he wants to know is the ductility, but it is difficult to ensure from one or two tests that the critical conditions for maximum embrittlement have been obtained. One advantage of very long-time rupture tests, apart from the fact that they need little attention, is that they provide an indication of the stress that can be withstood without rupture actually occurring. Dr. Bailey also points out the difficulties in extrapolating long-time creep tests on the basis of double logarithmic plotting and suggests that the results shown in Fig. 69 (*b*) of my second paper (p. 75) are too optimistic. In extrapolating the results I deliberately kept on the low side, but in view of the facts that the creep test carried out for 20,000 hours at a stress of 6 tons/sq. in. and a temperature of 550° C. had only extended 0·12%, and also that a rupture test at the same stress and temperature shows no sign of failure and has extended only a fraction of 1% after more than 70,000 hours under test, I feel justified in considering that the results are reasonably exact. Furthermore, I note from Fig. D that even after 1000 hours of tempering at 690° C. the steel does not show deterioration. Surely, then, after 5 hours' tempering at that temperature, the steel will continue to improve, over a very long period of time, at a lower temperature. Thus, in determining the true stress–temperature relationship, very prolonged long-time tests would be necessary to show up molybdenum–vanadium steel to full advantage.

In reply to Mr. Rotherham, I agree that in the comparison of short-time and long-time creep tests carried out by Johnson and Tapsell, a German test, and also the Hatfield time-yield test, place the steels in the same order as the long-time tests, but both these tests overestimated the poor creep resistance of the abnormal steel. If the tests had been carried out on a different set of steels with a wider variation of analysis and heat-treatment, it is probable that the short-time tests would have shown greater discrepancies. The reason for the choice of the test conditions used has been given in the reply to Mr. Bardgett, and is also detailed in the paper. Mr. Rotherham points out that there is close agreement between results for some of the molybdenum steels and some of the chromium–molybdenum steels. I agree that this is the case, but, actually, it is shown in the paper that results for these types of steels overlap. However, in similar conditions, *i.e.*, manufacture and heat-treatment, chromium–molybdenum steel is better than 0·5% molybdenum steel. I do not see why Mr. Rotherham thinks that chromium–molybdenum steel may show as much embrittlement as 0·5% molybdenum steel. Chromium–molybdenum steel, of course, does undergo embrittlement, but even when it has a creep resistance better than 0·5% molybdenum steel it can still show better ductility.

* *Journal of The Iron and Steel Institute*, 1947, vol. 155, April, pp. 501–512.

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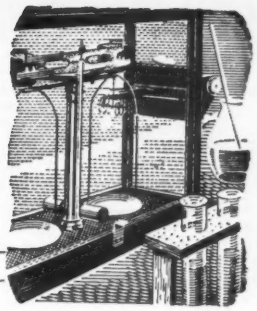

Ingot Structures in a Series from Rimmed to Killed Steels Made from the Same Cast

By P. M. Macnair, B.Sc., Ph.D.

SYNOPSIS

A series of ten ingots, ranging from box-hat, through rimming, and rising to killed steel, has been made from the same cast by additions of iron oxide and aluminium to the moulds. Vertical cross-sections through the centre-line were prepared and the ingots examined for blowhole appearance and chemical segregation of the major elements, carbon, sulphur, phosphorus, and manganese. A theory is advanced to explain the method of formation of the various types from wild to killed steel. This is based on two well-known principles, (1) selective freezing, (2) rate of reaction between free oxide and carbon.

Introduction

THE heterogeneity of steel ingots has been the subject of considerable investigations, and various suggestions have been advanced to explain the essential features of the structure produced in the ingot during solidification in the mould. The object of the present paper was to study a series from wild steel through box-hat, rimming, rising, balanced, and semi-killed to killed steel, all made from the same cast. The theoretical explanations which are given contain no fundamentally new principle but are rather an application of well-known principles to the solidification of a wide range of ingot types.

HISTORICAL SURVEY

While it is not intended to give a critical or extensive survey of the various papers which have been published, some of the ideas put forward towards the end of the nineteenth century are noted because these are the basis of the theoretical considerations given in this paper. According to Hultgren and Phragmen[1] it was stated that Walrand,[2] in 1882, "made the important observation that a crust without blowholes would solidify when the gas evolution was rapid, if only the gas bubbles were swept away by the boiling motion in the liquid metal. When the top of the ingot solidified, gas evolution was prevented by increasing pressure and the motion in the liquid came to an end.

"Walrand assumed the mould gas to be carbon monoxide forming by reaction between carbon and ferrous oxide dissolved in the molten steel. He supposed the oxygen content to increase when the carbon content decreased. The disappearance of the blowholes after sufficient additions of silicon was verified, and could easily be explained by the assumption that the silicon combined with the oxygen, thereby preventing the formation of carbon monoxide."

In 1892 Styffe[3] described the influence of aluminium additions. In ingots without additions, from a particular heat of steel containing 0·15% carbon, blowholes were located in a zone about 2 inches from the surface. Ingots from the same heat, to which 0·02% aluminium had been added in the mould, were honeycombed externally so as to be completely useless. Ingots with an addition of 0·04% aluminium were free from blowholes.

In 1907 Von Maltitz[4] assumed the formation of carbon monoxide on solidification of steel to be caused

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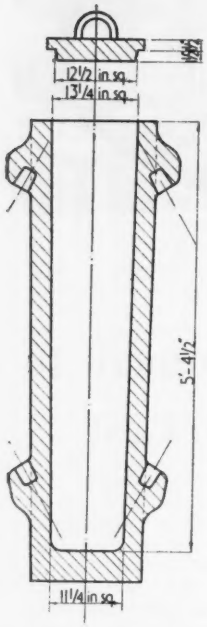


Fig. 1—Section of ingot mould and cover

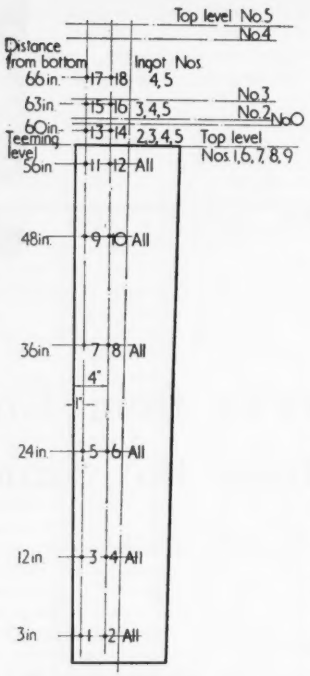


Fig. 2—Position in ingots for chemical analysis

by selective solidification. Carbon and ferrous oxide would accumulate in the still-liquid steel, the layers first solidified being purer than the original metal. When the concentration was sufficient, the mutual reaction would start.

PRESENT INVESTIGATION

The aim of the present investigation was to produce a series of 1-ton ingots from the same cast, varying only free oxygen, and keeping all the other factors as constant as possible. It was decided that the most promising method was to add predetermined quantities of iron ore and aluminium to successive moulds of a cast of rimming steel. The main advantage of this technique appeared to be the possibility of producing all types from the same cast of steel, with reasonable constancy in other known factors affecting solidification, such as temperature, rate of teeming, and size of mould. Once the series had been produced, the metallurgical examination was straightforward.

DETAILS OF EXPERIMENTAL WORK

STEELMAKING DATA

The steel was made at Duffryn Steelworks in a cold-charged, fixed-type basic open-hearth furnace. The charge consisted of 22 tons of pig iron, 32 tons of scrap, and 4 tons of limestone. About eleven hours after starting to charge, the furnace was clear melted off the bottom with a thick creamy slag and very good heat. The bath analysis was 0·33% carbon, 0·21% manganese, 0·32% phosphorus, 0·055% sulphur, with 7% iron in the slag. The charge was worked out with scale additions, a heavy one soon after clearing, and a lighter addition half an hour later. A sample drawn 15 min after the last addition showed 0·08% carbon, 0·14% manganese, 0·014% phosphorus, and 0·051% sulphur, with 12% of iron in the slag. It was decided to tap on this sample, and the tapping hole was opened 15 min later. The deoxidizing additions were 2 cwt of 80% ferro-manganese in the bath and 4 cwt in the ladle.

CASTING-PIT DATA

The pit analysis on an unkilled sample was 0·08% carbon, 0·055% sulphur, 0·018% phosphorus, 0·32% manganese, with a finishing iron of 13·2% in the slag. The cast was teemed through $1\frac{1}{4}$-in nozzles into 1-ton moulds in pairs. A section through the mould and cover is shown in Fig. 1. The experimental ingots were obtained starting from the thirteenth pair teemed, and additions were made only to the front ingot of the pair in succession from No. 13 to No. 22. In order not to interfere with teeming times no pit sample was drawn during this period, and to assist the teemer in filling all moulds to the same level, a guide mark was made by hanging on top of the mould a piece of bent steel strip, with a 1-in tongue sticking out into the mould at a depth of 6 in from the top. The ladle nozzle was open full bore from the start of each ingot, with only a slight easing off towards the end in order to shut off completely when the metal reached the guide mark.

It had been hoped to make a level rimmer, and additions had been prepared accordingly as 8 oz iron ore, 0, 3, 6, 9, 12, 16, 32, and 80 oz aluminium in succession. Observation on the first ingots teemed from the ladle showed the charge was a slightly rising rimmer, so an additional ingot was arranged for, after the last one of the series, to which 24 oz of iron ore were added. The ingot following this one was also retained, no addition being made, in order to get a comparison with the earlier ingot teemed on similar lines.

The steel in the moulds was allowed to react to its fullest extent, and there was no suppression by plating or covering. The time during which any top action on the ingot was taking place is recorded in Table I. The temperatures of the metal were taken during the pouring of the first and last ingots of the series, and were recorded as 1610° and 1600° C, respectively, after correction.

For identification purposes the ingots were numbered consecutively from 1 to 10; but as No. 10 was out of step in the series of decreasing oxygen, it was renumbered 0, by which numeral it is referred to throughout the paper.

The eleventh ingot, which had no addition, was compared with No. 2, which also had no addition. The amount of rise, as judged from the guide marks, and the general appearance of the two was so similar that No. 11 was discarded. It had served its purpose of showing that there had been no noticeable change in the average run of the metal from the ladle throughout the series.

Table I

EXPERIMENTAL RESULTS

Ingot No.	Ingot Type	Additions to Mould	Teeming Time, sec	Time before Top Solid, min	Weight of Ingot, cwt	Setting Action in Mould	Low Sulphur, Carbon, and Phosphorus Rim Thickness, in	Thickness of Solid Outer Skin, in		Primary Blowholes	Secondary Blowholes	Central Blowholes	Other Data
								Top	Bottom				
0	Box-hat	24 oz iron ore	50	15	17¼	Sink of 5 in, leaving box-hat. Rimmed. Formed cauliflower top	2	2 to secondaries	0·6	In bottom quarter only	Full length	Scattered throughout	Pit analysis (unkilled sample) C 0·08% S 0·055%
1	Level rimmer	8 oz iron ore	48	13½	18	Rimmed level with no rise or sink. No cauliflower top	2¾	2¾ to secondaries	0·5	In bottom half only	Full length	Scattered more particularly at top and bottom	P 0·018% Mn 0·32% Fe in slag 13·2%
2	Slightly rising rimmer	Nil	50	14	18½	Rise of 3½ in. Rimmed. No cauliflower top	2¾	2¾ to secondaries	0·35	In bottom half only	Full length	Scattered, more particularly at top and bottom	Teeming Level 58 in from base of ingot
3	Rising rimmer	3 oz Al	48	15	19½	Rise of 6 in. Rimmed. Formed cauliflower top reaching 11½ in. above teeming level	2¾	0·1	0·2	In bottom two-thirds only	Full length	Scattered throughout	Casting Temperature 1610–1600° C
4	Rising	6 oz Al	48	14½	19	Rise of 12¾ in. No apparent rimming. Formed cauliflower top	2¾	0·1	0·2	Almost full length	Full length	Scattered throughout	
5	(Fully) Rising	9 oz Al	47	13½	19¼	Rise of 14 in. No apparent rimming. Formed cauliflower top	2¾	0·1	0·2	Almost full length, and also at level at which no discontinuity	Only in top half	Only in top half	
6	Semi-killed	12 oz Al	50	Less than 10 sec	19½	Set flat at teeming level	Nil slight rim (¼ in thick at bottom)	Solid ingot		Nil	Nil	Only at top in pipe zone	
7 8 9	Killed	16 oz 32 oz 80 oz Al	50 48 48	Less than 10 sec	19¼ 19½ 19½	Set flat at teeming level	Nil	Solid ingot		Nil	Nil	Nil. Shrinkage cavity	

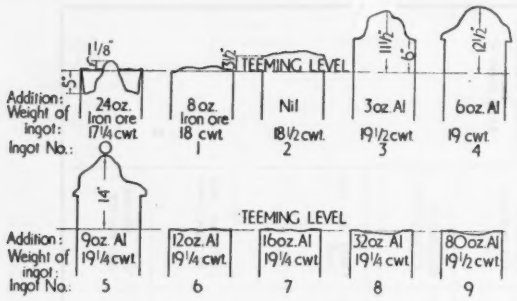


Fig. 3—Diagram of appearance of ingot tops

In all cases the additions were made in equal parts of one-third the previously estimated total quantity, when the mould was one-quarter, one-half, and three-quarters full.

PREPARATION OF THE INGOTS FOR EXAMINATION

When the ingots had cooled off, measurements were taken of the respective amounts of rise and sink from the guide mark, which had adhered to all ingots. Each ingot was weighed separately.

The ingots were then burned through, longitudinally, about 1 in off centre-line. The machined surface was photographed to show the blowhole appearance, and then sulphur printed. The remaining part of the burned ingots was broken across at roughly $\frac{1}{3}$ and $\frac{2}{3}$ the total length from the bottom. This was done to get confirmation of the depth of solid skin between the surface and the start of primary blowholes.

In order to observe the degree of heterogeneity of the elements, samples for analysis were taken from the same positions in all ten ingots; yet allowance had to be made for those ingots which had risen considerably. The points decided on are shown in Fig. 2. They are 1 in and 4 in, respectively, from the outside surface, and were selected to represent, (1) the rim at a point roughly half-way from the outer surface to the rim-core junction and, (2) the core, half-way between the rim-core junction and the centre of the ingot. In all ten ingots they were taken with a $\frac{1}{2}$-in drill at heights of 3, 12, 24, 36, 48, and 56 in from the base. Additional samples were taken, as shown, for ingots which rose above the 58-in teeming level. The elements determined were carbon, sulphur, phosphorus, and manganese. All portions of the ingots have been retained for further work.

EXPERIMENTAL RESULTS

Ten ingots ranging progressively from rimmed to killed steel were obtained. The appearance of the ingot tops is shown diagrammatically in Fig. 3, while a further impression is obtained from the photograph of all ten in Fig. 8. The machined surface, sulphur print, and the cracked sections, together with he names of each type, which are, of course, derived from their appearance, are illustrated for each ingot separately in Figs. 9 to 18. For ease of cross reference and comparison the experimental results are set out in Table I and only the main points are noted here.

The teeming times are practically constant throughout. The time that elapsed from the end of teeming till no further action was observed on top of the ingot was reasonably constant at around 15 min for each of the ingots in the range, from the box-hat to fully rising. The semi-killed and killed steels, however, set solid right across the top in less than 10 sec.

The weight of the ingots shows a definite and regular increase from No. 0 to No. 3, but is constant thereafter at 19¼ cwt. The column describing the setting action in the mould also gives the amount of sink or rise from the teeming level.

The thickness of the low carbon–sulphur–phosphorus rim is reasonably constant from the level-rimming to the fully-rising ingot, but is distinctly less in the box-hat ingot. In the killed steel there is, of course, no rim of this type. The sulphur print of the semi-killed steel, No. 6, shows about ½ in low sulphur at the bottom zone. It is felt that this is not significant and arises from one of the weaknesses of the experimental technique used—the time of the first aluminium addition. The addition of all the aluminium in one lump was considered to be too drastic, while it was feared that too early an addition might not be dispersed and cause formation of killed steel at the base. From the evidence of this ingot it is thought that the timing of the addition has a significance which may be as important as the total quantity.

The thickness of the outer solid skin down to the start of the primaries decreases from the box-hat to the rising rimmer, while the height to which the primaries extend from the base increases over this range. There is also a brief description of the appearance of the primary, secondary, and central blowholes in each ingot.

Where the ingot rose above the teeming level, there is no apparent discontinuity, either in the internal structure or the chemical analysis. It was confirmed, however, that there is the well-known alteration in the outside appearance, which allows an estimate of the amount of rise to be made. Ingots Nos. 4 and 5 show a marked and progressive decrease in the high-purity zone and length of primaries, starting somewhat below the teeming level and extending practically to the top.

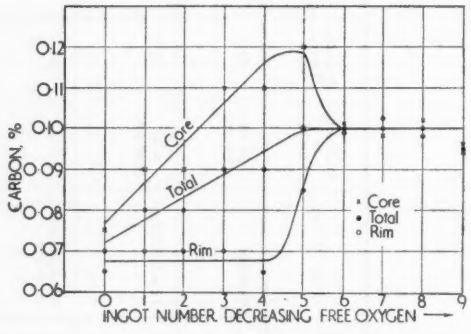


Fig. 4—Carbon variation in ingots 0–9, rim, core, and total

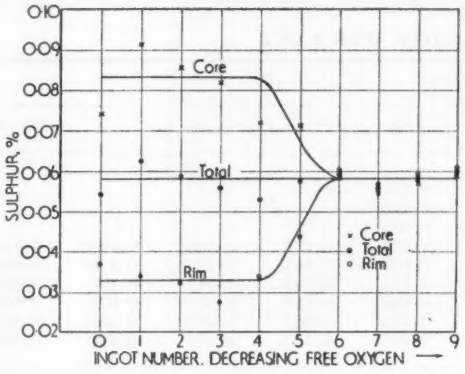


Fig. 5—Sulphur variation in ingots 0–9, rim, core and total

The standard points, at which carbon, sulphur, phosphorus and manganese were determined, are shown in Fig. 2, and the figures detailed in Tables II to V. The averages of each element in the rim, in the core, and the means of these two figures, are graphed in Figs. 4 to 7 for each ingot.

OTHER EXPERIMENTAL WORK

Experiments, which had been previously carried out but not reported, seemed to throw some further light on blowhole formation and, consequently, the opportunity has been taken to incorporate them in the present paper. They were arranged to observe (1) the effect of suppression of gas evolution on primary blowhole formation in rising steel, (2) the effect of suppression of gas evolution on primary blowhole formation in rimming steel, and (3) the effect of casting an ingot with the mould tilted.

Effect of Suppression of Gas Evolution (Rising Steel)

Procedure—The steel was made at Duffryn Steelworks under similar conditions to the first experiment, using the same sized moulds. In order to produce a rising steel, there was an extra ladle addition of 50 lb of 47% ferrosilicon. The pit analysis was 0·09% carbon, 0·042% sulphur, 0·019% phosphorus, 0·39% manganese.

Four successive ingots were teemed to varying levels, using guide strips, so as to allow rises of $7\frac{1}{2}$, $4\frac{1}{2}$, $1\frac{1}{2}$, and 0 in to the inside of the cover. On completion of teeming the covers were put on top of the moulds and, therefore, no action was seen except for pale blue flames from the sides of the covers.

Results—All ingots rose cleanly to the covers and stripped with them.

The sulphur prints and cracked sections are shown in Figs. 19, 20, and 21. There is no significant variation in the depth of solid skin before primary-blowhole formation starts, but there is a definite and progressive change in the length of the primary blowholes, which decreases from 1·7 to 1·0, 0·9, and 0·6 in, respectively, from the greatest to the least rising ingot. This is considered to be evidence in favour of the suggestion that primary-blowhole formation is stopped by the ingot solidifying over on the top.

The ingot which was allowed to rise $7\frac{1}{2}$ in is felt to be of further interest as it probably helps to fill in the gap between ingots Nos. 5 and 6 in the original 10-ingot series. It did not rise fully, even at $7\frac{1}{2}$ in, as can be seen from the flat top where it was forced against the cover.

There is no primary-blowhole formation on the base, or for the first twelve inches up from the bottom. Primary-blowhole formation stretches from this point to the top of the ingot. There are no secondary or central blowholes except for the scattered range of centrals at the top, which is the main feature of ingot No. 6.

The sulphur print shows a high-purity zone associated with the primaries and a rather impure zone at their inward end. The base and the bottom-solid zone of 12 in appear to be more even in composition from edge to edge of the ingot than higher up. There is also slight evidence of the less-pure zone tapering off to the edge from the lowest point of the primary blowholes towards the base of the ingot.

Effect of Suppression of Gas Evolution (Rimming Steel)

Procedure—The steel was made under similar conditions to those reported for the ten ingots, and gave a pit analysis of 0·085% carbon, 0·040% sulphur, 0·020% phosphorus, 0·32% manganese. Three successive ingots were (1) allowed to rim out without

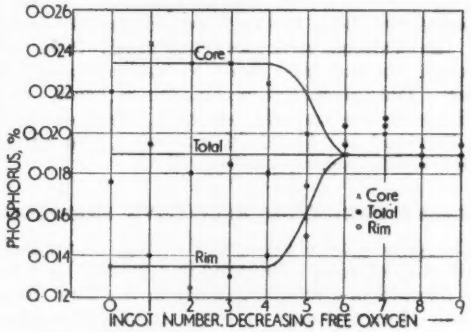


Fig. 6—Phosphorus variation in ingots 0–9, rim, core, and total

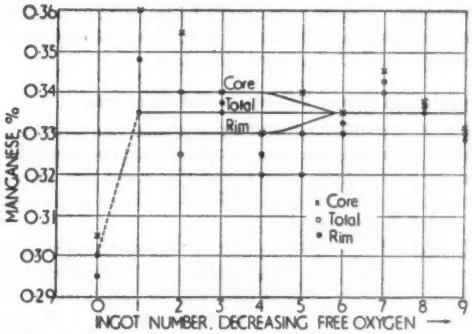


Fig. 7—Manganese variation in ingots 0–9, rim, core, and total

Table II

CARBON CONTENTS OF INGOTS (PER CENT)

Position in Ingot	Ingot No.									
	0	1	2	3	4	5	6	7	8	9
Rim										
17	...	...	...	...	(0·135)	0·110	...	...	...	...
15	...	...	...	(0·075)	0·065	0·115	...	...	...	...
13	...	...	0·060	0·075	0·065	0·080	...	...	...	...
11	Cavity	0·065	0·065	0·060	0·060	0·075	0·105	0·110	0·100	0·120
9	0·070	0·065	0·070	0·065	0·060	0·070	0·105	0·115	0·110	0·095
7	0·065	0·065	0·075	0·065	0·060	0·075	0·105	0·115	0·100	0·085
5	0·070	0·075	0·075	0·075	0·070	0·060	0·100	0·100	0·075	0·075
3	0·065	0·070	0·065	0·075	0·070	0·090	0·080	0·085	0·085	0·090
1	0·060	0·070	0·070	0·065	0·080	0·085	0·120	0·105	0·100	0·105
Average	0·065	0·068	0·068	0·069	0·066	0·084	0·102	0·105	0·095	0·095
Core										
18	...	...	...	...	0·120	0·145	...	...	...	...
16	...	...	...	0·125	0·130	0·145	...	...	...	...
14	...	...	0·095	0·115	0·100	0·155	...	...	...	...
12	Cavity	0·115	0·100	0·110	0·125	0·120	0·135	0·100	Cavity	Cavity
10	0·110	0·100	0·100	0·095	0·100	0·120	0·100	0·110	0·100	0·105
8	0·100	0·100	0·090	0·115	0·115	0·115	0·105	0·115	0·105	0·100
6	0·065	0·085	0·090	0·105	0·110	0·090	0·095	0·090	0·105	0·090
4	0·055	0·080	0·090	0·110	0·105	0·110	0·070	0·100	0·105	0·085
2	0·055	0·065	0·080	0·095	0·070	0·095	0·105	0·085	0·100	0·080
Average	0·077	0·091	0·092	0·109	0·108	0·122	0·102	0·100	0·103	0·094
Average Rim and Core	0·071	0·080	0·080	0·089	0·091	0·103	0·102	0·102	0·099	0·059

any suppression, (2) plated when the rim had formed about 3 in, (3) fully suppressed by placing a cover on top of the molten metal in the mould shortly after completion of teeming.

Results—The sulphur print of each ingot is shown in Fig. 22, and the appearance of the ingot tops and the cracked transverse sections in Fig. 23.

Although the ingots contain a number of interesting features, it is desired only to call attention to the skin thickness and the primary-blowhole length. Primaries occur only at the bottom of the ingots, and all three show a constant skin thickness of about 0·4 in. The length of primary blowholes, however, decreases progressively from 1·65 to 1·6 and 1·35 in respectively. This is considered to be further confirmation that primary-blowhole formation is stopped by the ingot solidifying over the top.

Experiment on the Effect of a Tilted Mould

Procedure—An ingot of similar analysis made under similar conditions was teemed into a tilted mould, so that one top-edge was vertically over the opposite bottom-edge. The metal rimmed level at the point to which it was teemed.

Result—The sulphur print of the sectioned ingot is shown in Fig. 24, and there are no outstanding differences from those which can be anticipated from similar steel cast in a vertical mould. Primary-blowhole formation appears to be at right angles to the mould wall, which indicates that the rate of inward solidification is still the major factor. The evenness of the depth of rim, equally on the upper and lower sides, also suggests that inward solidification is more important than any effects of gas evolution. In considering the theory of solidification of steel ingots it is therefore felt that, while gas evolution carries away molten mother liquor, it has practically no effect on the advancing solid wall.

THEORETICAL CONSIDERATIONS

There has been considerable discussion on the fundamental principles governing the heterogeneity produced in steel ingots. These suggestions have been examined further from the viewpoint of the present experiments and a theory is put forward to account for the various structures occurring, from wild, through rimming, and rising to balanced and killed steels. The theoretical explanation is based on two well-known fundamental principles, which can briefly be described as : (i) selective freezing, and (ii) rate of free-oxide/carbon reaction.

FIRST PRINCIPLE: SELECTIVE FREEZING

The principles of selective freezing, as applied to a killed steel ingot solidifying in a mould, have been put forward in a simple manner in the First Report of the Heterogeneity of Steel Ingots Committee.[5] This report gives an adequate and comprehensive explanation of the major segregation which takes

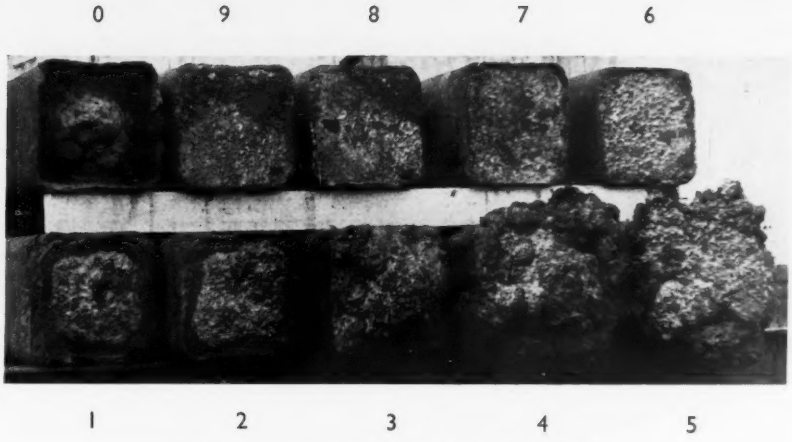


Fig. 8—Appearance of ingot tops. Ingots 0-9

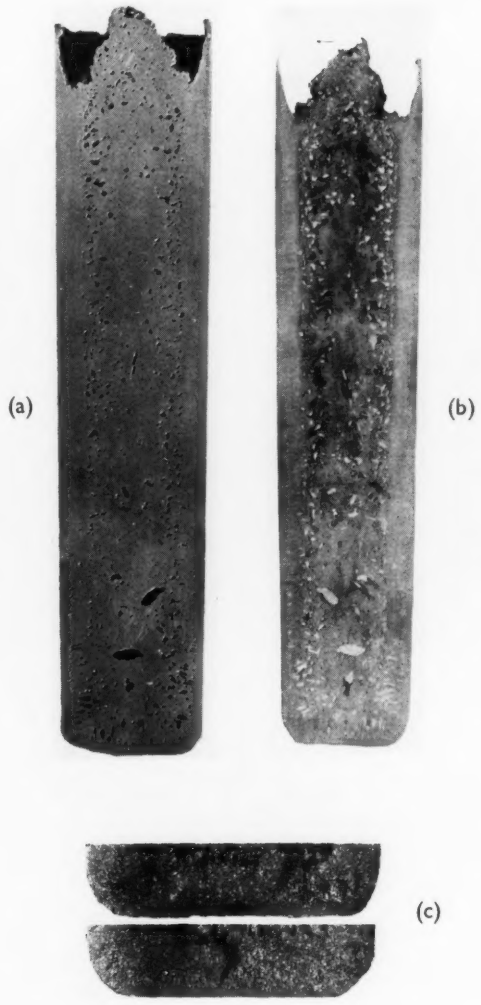


Fig. 9—Ingot 0, box hat

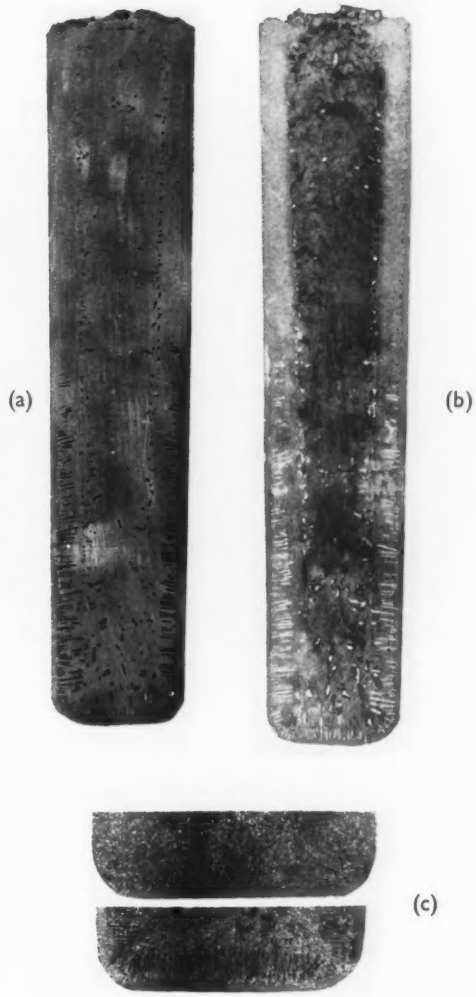
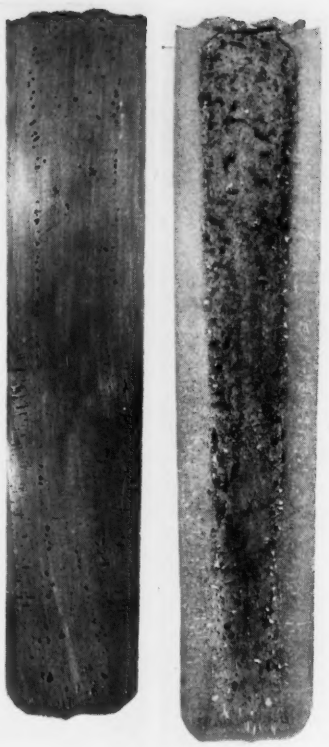
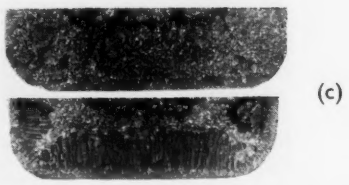


Fig. 10—Ingot 1, level rimmer

Figs. 9 and 10—(a) Machined surfaces; (b) sulphur prints; and (c) fracture samples



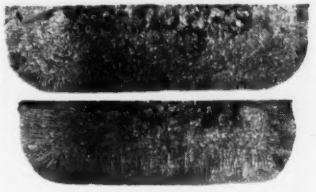
(a) (b)



(c)

Fig. 11—Ingot 2, slightly rising rimmer

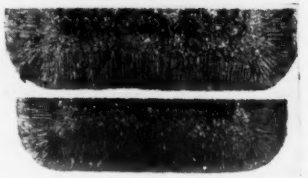
(a) (b)



(c)

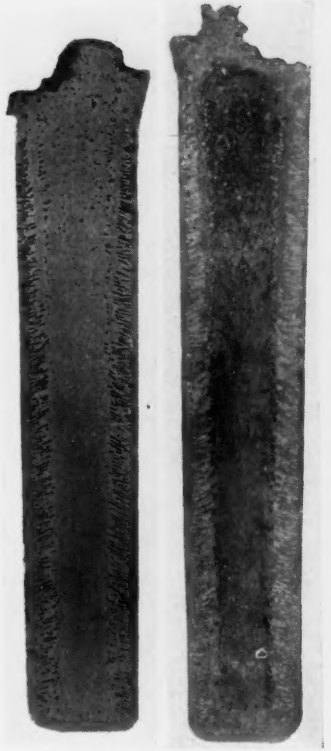
Fig. 12—Ingot 3, rising rimmer

(a) (b)

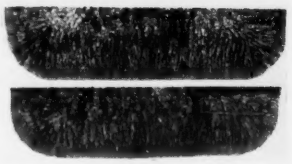


(c)

Fig. 13—Ingot 4, rising



(a) (b)



(c)

Fig. 14—Ingot 5, full y rising

Figs. 11 to 14—(a) Machined surfaces ; (b) sulphur prints ; and (c) fracture samples

Fig. 11—Ingot 2, slightly rising rimmer

Fig. 12—Ingot 3, rising rimmer

Fig. 13—Ingot 4, rising

Fig. 14—Ingot 5, fully rising

Figs. 11 to 14—(a) Machined surfaces ; (b) sulphur prints ; and (c) fracture samples

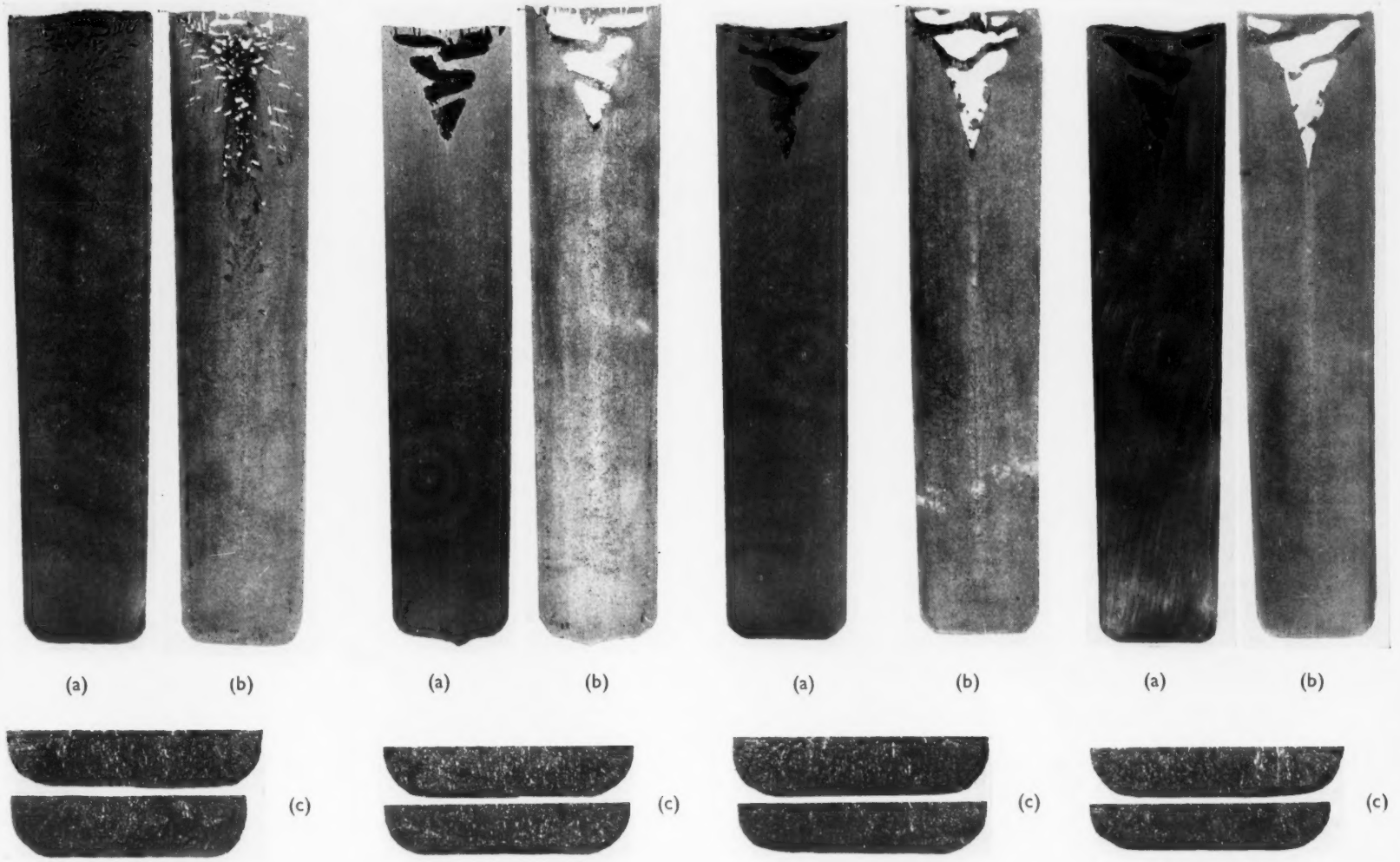


Fig. 15 –Ingot 6, semi-killed

Fig. 16—Ingot 7, killed

Fig. 17—Ingot 8, killed

Fig. 18—Ingot 9, killed

Figs. 15 to 18—(a) Machined surfaces ; (b) sulphur prints ; and (c) fracture samples

Macmair

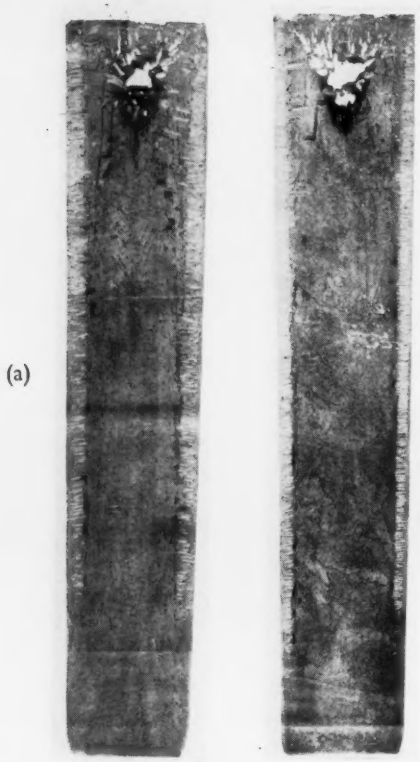


Fig. 19—Sulphur prints of ingots rising : (a) $7\frac{1}{2}$ in. and (b) 4 in.

Fig. 20—Sulphur prints of ingots rising : (c) $1\frac{1}{2}$ in. and (d) 0 in.

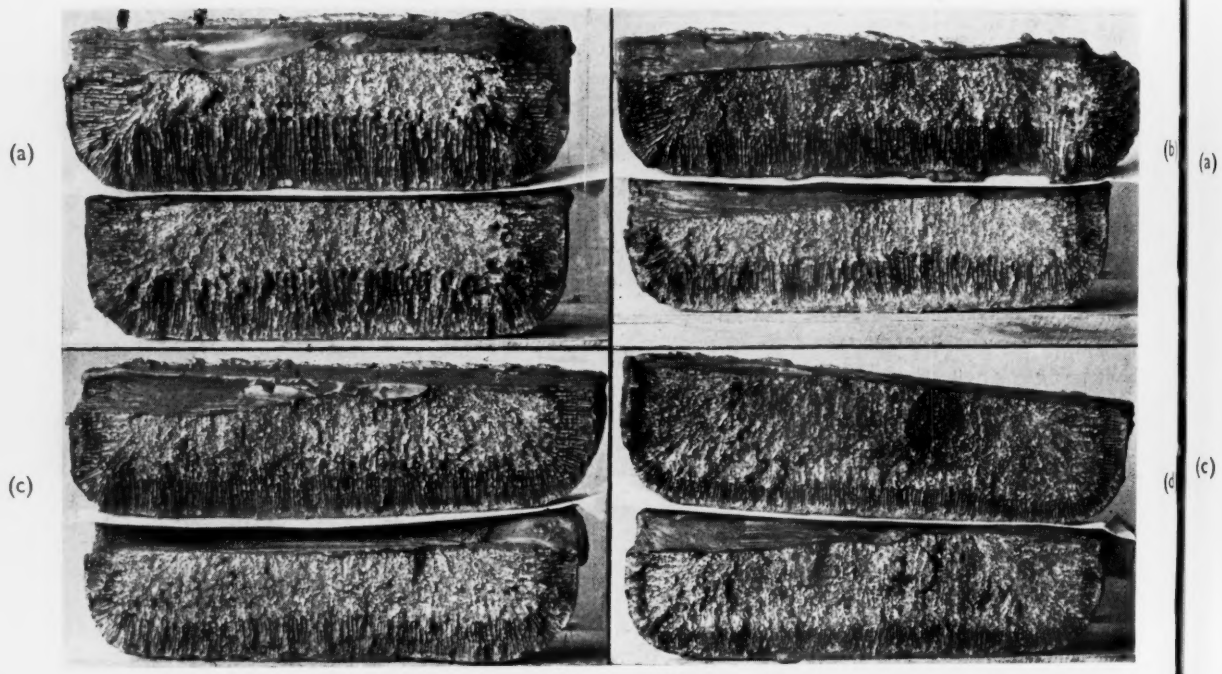


Fig. 21—Fracture samples of ingots shown in Figs. 19 and 20

Figs. 19 to 21—Effect of suppression of gas evolution on primary blow-hole formation in rising steel

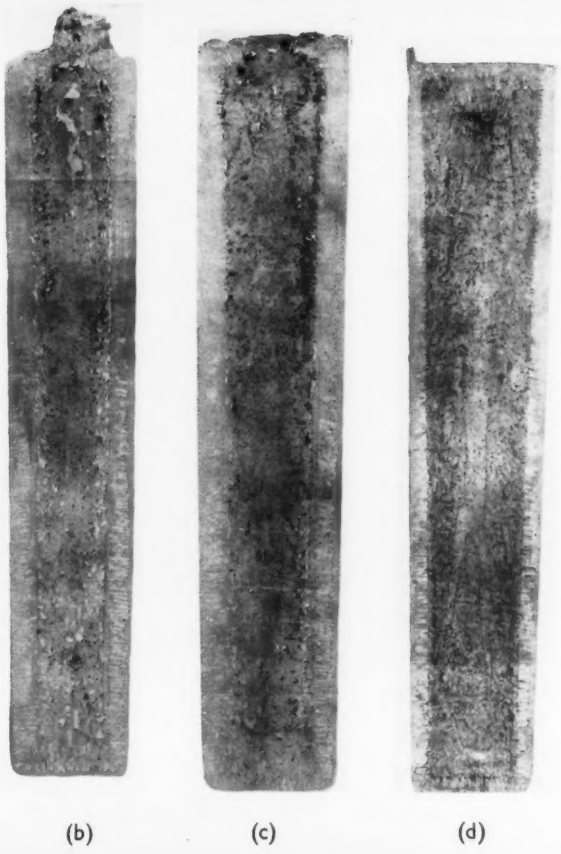


Fig. 22—Sulphur prints of ingots shown in Fig. 23

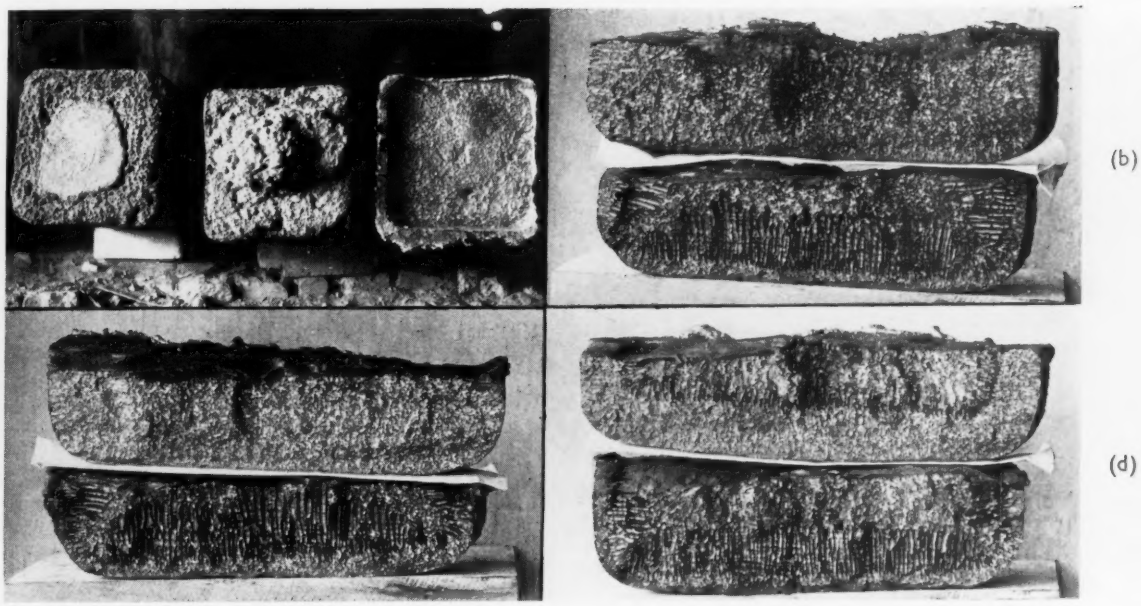


Fig. 23—Test No. 3. (a) Appearance of ingot tops. (b)–(d) Transverse fracture samples of rimming steel ; (b) no suppression ; (c) plated ; and (d) fully suppressed

Figs. 22 and 23—Effect of suppression of gas evolution on primary blow-hole formation in rimming steel

Fig. 24—Sulphur print of ingot cast in tilted mould

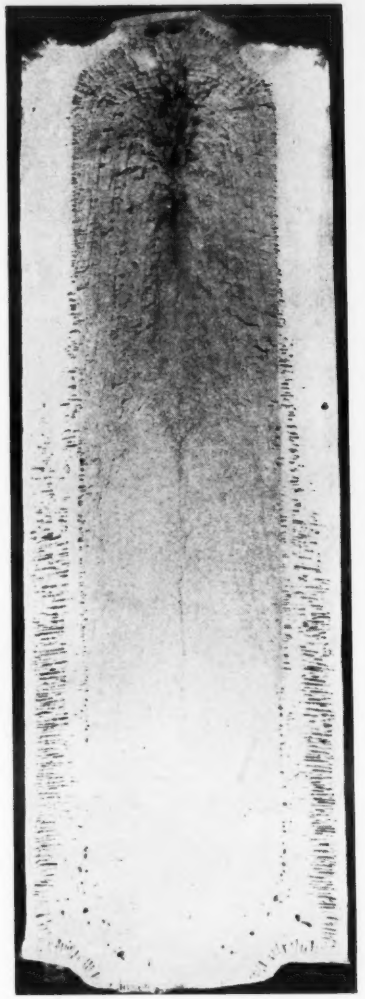


Fig. 25—Sulphur print of 8-ton rimming-steel slab ingot, showing primary blow-holes tapering-off at top

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Table III
SULPHUR CONTENTS OF INGOTS (PER CENT)

Position in Ingot	Ingot No. 0	1	2	3	4	5	6	7	8	9
Rim										
17	...	...	...	...	(0·121)	0·041	...	...	...	...
15	...	...	...	(0·043)	0·033	0·044	...	...	...	...
13	...	...	0·040	0·025	0·034	0·039	...	...	...	...
11	Cavity	0·033	0·028	0·025	0·030	0·043	0·051	0·056	0·058	0·058
9	0·033	0·033	0·034	0·029	0·031	0·045	0·059	0·059	0·058	0·060
7	0·033	0·034	0·036	0·029	0·033	0·042	0·051	0·053	0·058	0·063
5	0·036	0·040	0·030	0·028	0·035	0·042	0·063	0·058	0·060	0·058
3	0·039	0·035	0·030	0·028	0·035	0·050	0·075	0·055	0·060	0·064
1	0·035	0·030	0·030	0·031	0·042	0·052	0·060	0·057	0·058	0·060
Average	0·0352	0·0342	0·0326	0·0280	0·0341	0·0442	0·0598	0·0563	0·0586	0·0605
Core										
18	...	...	...	...	0·099	0·098	...	...	...	...
16	...	...	...	0·143	0·099	0·087	...	...	...	...
14	...	...	0·107	0·106	0·086	0·068	...	...	...	...
12	Cavity	0·152	0·122	0·083	0·065	0·071	0·077	0·053	Cavity	Cavity
10	0·088	0·110	0·088	0·072	0·075	0·068	0·052	0·057	0·058	0·057
8	0·138	0·107	0·101	0·085	0·080	0·067	0·057	0·056	0·055	0·064
6	0·059	0·070	0·089	0·066	0·062	0·055	0·054	0·059	0·059	0·060
4	0·054	0·064	0·055	0·063	0·038	0·071	0·061	0·056	0·058	0·058
2	0·031	0·042	0·035	0·040	0·044	0·052	0·053	0·050	0·055	0·060
Average	0·074	0·0908	0·0853	0·0822	0·0720	0·0710	0·0590	0·0552	0·0570	0·059
Average Rim and Core	0·0564	0·0625	0·0590	0·0560	0·0579	0·0576	0·0594	0·0557	0·0578	0·0601

place when killed steel freezes. The essential ideas may be summarized as follows :

(*A*) A very thin outer "chill" layer is formed by the almost instantaneous freezing of the molten steel when it comes into contact with the mould wall, giving only very minute crystals, which are of the same composition as the liquid steel.

(*B*) Progressing inwards, the region of columnar crystals, which have their major axis perpendicular to the mould wall, is next found. Crystallization, while still rapid, is slow enough to be selective and the cores of the solid crystals are purer than the original steel.

(*C*) The liquid steel existing in proximity to the already frozen columnar region becomes richer in "impurities" than the original liquid steel, and this is shown by a rather higher concentration towards the junction of the columnar and equiaxed zones.

(*D*) Following the formation of the columnar zone, equiaxed crystals of relative purity begin to form in the central portion of the ingot and fall towards the bottom, thus giving a bottom zone of comparatively high purity. The impure mother liquor, rising in the central portion, solidifies with increasing segregation towards the top central zone.

The terms "impure" and "impurity" are used throughout the report for all elements other than iron, irrespective of whether their presence is desired in the steel.

SECOND PRINCIPLE: RATE OF FREE-OXIDE/CARBON REACTION

While there has been considerable controversy over the composition of the gases which are evolved when any steel other than killed is made, there is a fairly general recognition that the free-oxide/carbon reaction to give CO, or more correctly a mixture of CO and CO_2, is the chief source of such gases. The extent to which this reaction takes place determines the type of ingot produced, and one of the most significant aspects to be considered is the rate at which the reaction is proceeding at any particular time.

For convenience three different rates of reaction are proposed to explain the structures produced :

(*A*) A high rate of reaction, where gas evolution is so great that no blowholes are formed. All the gases and the impure mother liquor produced by selective freezing are swept away from the advancing solid wall.

(*B*) A lower rate of reaction, such that some bubbles of gas adhere to the advancing solidifying wall, thus forming nuclei on which blowhole formation proceeds. Some quantities of gas, however, are still swept away as in (*A*), and carry the mother liquor with them. This gives blowholes of the primary type.

(*C*) A very low rate of reaction, where blowholes are formed, but all of the gas remains *in situ*. This occurs most generally when the steel is in a "mushy" state, and gives rise to the secondary and central types of blowhole.

There is, in addition, the condition where the rate of reaction is zero, and no gas bubbles or blowholes are formed.

Table IV

PHOSPHORUS CONTENTS OF INGOTS (PER CENT)

Position in Ingot	Ingot No.									
	0	1	2	3	4	5	6	7	8	9
Rim										
17	...	...	...	...	(0·033)	0·014	...	...	...	...
15	...	...	...	(0·019)	0·015	0·015	...	...	...	...
13	...	...	0·013	0·011	0·015	0·015	...	...	...	...
11	Cavity	0·014	0·011	0·012	0·012	0·016	0·021	0·021	0·020	0·020
9	0·011	0·013	0·013	0·015	0·012	0·016	0·021	0·021	0·019	0·019
7	0·013	0·014	0·014	0·015	0·014	0·016	0·023	0·019	0·019	0·018
5	0·014	0·014	0·021	0·014	0·016	0·012	0·019	0·020	0·017	0·020
3	0·014	0·015	0·013	0·011	0·014	0·015	0·018	0·022	0·018	0·020
1	0·015	0·015	0·012	0·012	0·015	0·016	0·020	0·021	0·018	0·019
Average	0·0134	0·0142	0·0126	0·0129	0·0141	0·0150	0·0203	0·0206	0·0185	0·0193
Core										
18	...	...	...	...	0·028	0·026	...	...	...	...
16	...	...	...	0·034	0·028	0·022	...	...	...	...
14	...	...	0·027	0·030	0·028	0·021	...	...	...	...
12	Cavity	0·035	0·029	0·023	0·019	0·024	0·022	0·019	Cavity	Cavity
10	0·027	0·029	0·026	0·020	0·020	0·024	0·019	0·022	0·020	0·020
8	0·028	0·025	0·025	0·024	0·022	0·020	0·020	0·020	0·019	0·019
6	0·020	0·023	0·024	0·021	0·020	0·010	0·017	0·021	0·022	0·017
4	0·020	0·018	0·019	0·020	0·020	0·017	0·018	0·020	0·016	0·018
2	0·014	0·017	0·015	0·015	0·018	0·015	0·019	0·019	0·021	0·0184
Average	0·0218	0·0245	0·0236	0·0234	0·0225	0·0199	0·0192	0·0202	0·0196	0·0184
Average Rim and Core	0·0176	0·0193	0·0181	0·0185	0·0191	0·0175	0·0197	0·0204	0·0190	0·0188

Carbon–Oxygen Reaction

The second principle is based on the rate of reaction between carbon and oxygen, and it is necessary to examine this reaction in some detail. It is generally expressed in the form

$$FeO + C \rightleftarrows CO + Fe$$

and the rate of reaction, with the usual notation, as

$$\frac{dx}{dt} = k_1 [FeO] [C] - k_2 P_{CO}$$

While there is much controversy over slag–metal equilibrium in steel-making processes, there is evidence that there is a considerable degree of equilibrium within the metal phase prior to teeming. The author believes that the metal phase can be taken as in equilibrium when it leaves the ladle for filling the moulds and therefore :

$$k_1 [FeO] [C] = k_2 P_{CO}$$

This gives a datum line from which to examine what is taking place during solidification of the ingot. It is realized that there is certain to be a difference of opinion as to whether the liquid steel phase can be considered to be in equilibrium. Although it appears to make the argument a little tidier, the point is not of major importance to the theory as the suggestions to be put forward indicate that the changes in concentration or pressure will " increase " the reaction rate in one direction or the other. That is the essential point. As a consequence of accepting equilibrium, it is necessary to examine the factors which may " restart " the reaction.

Effect of Selective Freezing

This is the main fundamental principle which accounts for the heterogeneity encountered in steel ingots. When solidification starts, the composition of the initial skeleton crystallites is lower in all elements, except iron, than the liquid from which they are forming. It follows that the concentration in the mother liquor, immediately adjacent to the solidifying wall, must increase. Although it is very difficult to get an estimate of the increased local concentration in the liquid, it is perhaps not so difficult to visualize that it is sufficiently great to upset equilibrium. While this would tend to restart the oxidation of all elements, the carbon–oxygen reaction is of outstanding importance at this stage because it causes the formation of a new phase in the form of a gas. It is suggested that gas evolution of CO is restarted, or increased, by the increase in the term [C][Feo], arising from local concentration in the mother liquor, due to selective freezing.

One other important aspect may be mentioned. Except for a very short time during the initial chilling against the mould wall, selective freezing is going on throughout the whole period of solidification. Accordingly reaction is liable to continue throughout the

Table V

MANGANESE CONTENTS OF INGOTS (PER CENT)

Position in Ingot	Ingot No.									
	0	1	2	3	4	5	6	7	8	9
Rim										
17	...	...	...	...	0·33	0·30	...	...	...	...
15	...	...	...	0·33	0·31	0·31	...	...	...	...
13	...	...	0·31	0·32	0·31	0·32	...	...	...	...
11	Cavity	0·33	0·32	0·34	0·33	0·32	0·33	0·35	0·35	0·34
9	0·30	0·33	0·33	0·34	0·32	0·32	0·34	0·35	0·34	0·33
7	0·29	0·34	0·33	0·34	0·31	0·32	0·34	0·34	0·33	0·33
5	0·30	0·34	0·33	0·33	0·33	0·32	0·33	0·33	0·34	0·33
3	0·30	0·34	0·33	0·33	0·32	0·33	0·33	0·33	0·34	0·33
1	0·29	0·34	0·33	0·33	0·33	0·34	0·30	0·33	0·32	0·33
Average	0·296	0·337	0·326	0·333	0·321	0·320	0·328	0·338	0·337	0·332
Core										
18	...	...	...	...	0·33	0·34	...	...	...	...
16	...	...	...	0·39	0·37	0·33	...	...	...	...
14	...	...	0·38	0·36	0·31	0·33	...	...	...	...
12	Cavity	0·39	0·37	0·32	0·33	0·34	0·35	0·34	Cavity	Cavity
10	0·32	0·36	0·36	0·33	0·32	0·34	0·34	0·35	0·34	0·34
8	0·33	0·37	0·36	0·34	0·31	0·33	0·34	0·35	0·33	0·33
6	0·30	0·36	0·35	0·33	0·33	0·34	0·33	0·34	0·33	0·33
4	0·29	0·36	0·34	0·33	0·32	0·35	0·33	0·34	0·33	0·33
2	0·29	0·34	0·33	0·34	0·34	0·34	0·33	0·34	0·34	0·33
Average	0·306	0·363	0·356	0·339	0·329	0·358	0·337	0·343	0·334	0·332
Average Rim and Core	0·301	0·350	0·341	0·336	0·325	0·329	0·332	0·340	0·335	0·332

whole period, and is not limited to any particular time or zone.

Free Oxygen and Its Effect

In the equation representing the reaction between oxygen and carbon in steel the concentration of oxygen is generally written as [FeO], while the carbon is shown as [C]. It is certain that neither oxygen nor carbon exist in the elemental state in molten steel. They can be considered to be in solution as separate entities, but it is more likely that they exist as chemical compounds. With present-day knowledge it does not appear to matter much whether the carbon is combined solely with iron or partly with one of the other elements, and so it is written as [C]. With oxygen, however, the position is different, and the fact that oxygen concentrations are written as [FeO] has a considerable significance.

It is generally agreed that total oxygen is not a fully significant figure when considering solidification. The oxygen or oxide which reacts with the carbon is more usually described as "free," "reactive," "dissolved," or simply "ferrous oxide." The present experiment gives some further confirmation of the desirability of a modifying term. Oxygen determinations are being undertaken by the Gases and Non-metallics Sub-Committee, and these will be reported later. It is hoped they may throw more light on the question of "free" oxygen, but they cannot be anticipated at this stage.

The oxygen that is not "free" is that which is combined in forms from which it is not readily reduced by carbon under the existing conditions, *e.g.* alumina and possibly some complex silicates.

The use of the term ferrous oxide or FeO to describe the "free" oxygen is probably not strictly correct from the viewpoint of the law of mass action, but it can be accepted as a reasonable first approximation.

Accordingly the reaction as written gives an indication of the kinetics and equilibria of the oxygen–carbon–iron system, even in the presence of other elements.

While it is suggested in the last section that it is the local increase in carbon at the solidifying wall which restarts the carbon–oxygen reaction, it is thought that the total free oxygen content plays the major rôle in controlling the rate of reaction.

When dealing with a practically constant analysis, as is the case in the present series, it seems reasonable to assume that the carbon concentration due to selective freezing is the same at the initial stages throughout the series. Rate of reaction, however, is dependent on the product of [C] and [FeO], so that for steel of any given analysis, varying only in free oxygen content, the initial rate of gas evolution increases accordingly as the free oxygen increases, and *vice versa*. This was clearly demonstrated in the experiment by the conversion of free oxygen to combined oxygen by means of aluminium additions.

As reaction proceeds the effect of decrease in

oxygen is generally more marked than decrease in carbon. For each 0·01% by weight of oxygen removed, there is 0·0075% of carbon evolved from the solidifying ingot. In the majority of commercial steels the carbon content is over 0·06% and the total oxygen content less than 0·04%, while the free oxygen is less than this latter figure. It is evident that, as the carbon and oxygen are removed in the above ratios, the concentration of oxygen in the remaining liquid must decrease relatively more rapidly than the concentration of carbon.

Therefore, as the rate of reaction is dependent on the product of the two concentrations the oxygen content has a greater effect on the rate of gas evolution.

While this condition applies to the present series, and to the vast majority of steels produced commercially, it is not without interest to note that it is possible for conditions to arise where [C] would be the controlling factor. This could probably be the position when carbon had dropped to 0·03% with a low degree of deoxidation in the ladle, when the carbon would appear to be removed relatively faster than the oxygen.

Effect of Pressure

The reaction between free oxygen and carbon is dependent on pressure, as there is one gaseous molecule on the right-hand side of the equation and none on the left. With the extra ferrostatic pressure at lower parts in the liquid interior of the ingot, the rate of back reaction increases, and this slows up the total effective forward rate of reaction which has been discussed earlier.

During the initial stages of solidification in the mould, it seems reasonable to assume that the factors causing increase in the forward rate of reaction are practically constant throughout the total length of the solidifying wall. The back reaction rate, however, increases with increasing depth of the ingot, and would be twice as great at a depth of about 57 in, as at the surface. The total effect of the two factors is to produce a decreasing rate of gas evolution at greater depths in the molten ingot.

With solidification of the top surface of the ingot, the back reaction pressure can build up to sufficient values to neutralize the forward reaction at all levels, and thus produce a state of equilibrium. Some evidence supporting this was obtained in the experiments on the effect of suppression of gas evolution on primary-blowhole formation, described earlier.

APPLICATION OF THEORY TO PRESENT SERIES

The internal structure of any ingot is dependent on many factors other than the simple free oxygen content, even if the latter is thought to be of major significance. It is therefore proposed to apply the theory to the present series, which illustrates the effect of varying the free oxygen content on the solidification of a 1-ton ingot of 0·10% carbon steel. The subject matter is dealt with in two parts ; in the first blowhole formation is considered in a continuous sequence from wild, through rimming, rising, balanced, and semi-killed to killed steel ; in the second, the segregation of the elements is considered.

BLOWHOLE FORMATION

Wild Steel

In wild steel there is the maximum free oxygen and the maximum rate of gas evolution. It is not a commercial steel and its production is rather a matter of theoretical interest. The only evidence about it from the present series is by extrapolation from the box-hat ingot (No. 0), and the main supporting evidence is obtained from the series of small ingots produced by Edwards.[6]

On completion of teeming the ingot is frothed up in the mould. The initial rate of reaction is so high and the gas evolution so great that there is no blowhole formation during the initial setting of the ingot (2/*A*).* As oxygen is used up and evolved as CO, the rate of reaction decreases, the frothiness decreases, and the liquid metal level falls. The rate of reaction is still so high, however, that all gases are swept away and there is no blowhole formation in the solidified metal. In extreme cases the liquid metal may sink practically to the bottom leaving only a more or less solid shell. In less extreme cases, with lower initial free oxygen, a gradually increasing quantity of mushy metal is left at the base in which large globular holes are formed (2/*C*).

Box-Hat Steel

Coming down the scale of free oxygen from wild steel, the first type represented in the series is the box-hat ingot (No. 0). The first metal to set is the chilled skin (1/*A*), which is followed by ingrowing dendrites (1/*B*). The steel is frothed up, as shown by the ingot weight, and this is accounted for by the high rate of gas evolution arising from the solidification already occurring in the lower part of the ingot. The rate is so high (2/*A*) that a solid skin is forming, and all gas escapes from the inward solidifying wall. At the same time the gas sweeps away the mother liquor of dendrite formation and thus a zone low in carbon, sulphur, and phosphorus is formed.

As the rate of reaction decreases due to oxygen loss as CO, the level of the liquid metal sinks, and the box-hat is formed. The rate of gas formation, however, is still sufficiently intense to form a solid skin throughout the whole length of the ingot.

With a further drop in reaction rate due to further oxygen loss there comes a time when gas bubbles begin to adhere to the steel (2/*B*), and primary blowholes start to form. The fact that the blowholes have their characteristic elongated appearance at right angles to the mould wall is no doubt connected with the quite rapid rate of solidification (1/*B*). The primary blowholes start to form at the bottom end of the ingot, both from the base and the sides, because at this zone the back reaction rate due to pressure has its highest value and thus its greatest effect in reducing the total forward reaction rate.

With the progressive lowering of the reaction rate,

* Reference to items in the First and Second Principles are referred to in the text as (1/*A*), (2/*C*), etc., inferring Principle (1/*A*), (2/*C*), etc., (*see* p.157).

the primary blowholes start to form (*i*) at greater heights in the ingot, and (*ii*) at greater depths inward from the outside wall. The former effect arises from the decreasing difference between the forward and back reactions at all levels in the ingot, and the latter from the time factor in solidification. This is not well illustrated in any of the ingots of the present series, but is clearly shown in the 8-ton ingot (*see* Fig. 25). These larger ingots show more detail than smaller ones, due to the longer time factor in solidification.

As already stated (2/*B*), while primary-blowhole formation is proceeding, the rate of reaction is still sufficiently great to sweep away mother liquor and leave a relatively pure solid zone.

Before dealing with the effect of the final closing over of the top of the ingot, the chilling effect on the top surface may be noted. It takes place in a similar way to bottom and side chilling, but is less severe in rimming steels, due to circulation of molten metal. The combined effect of top, and side, cooling causes a rounding-off of the purer rim zone at the top of the ingot. The zone is free from blowholes owing to high rate of reaction with low back pressure.

The primary-blowhole formation and segregation proceed in this pattern until the top surface of the ingot solidifies over, which occurs while there is still molten metal in the interior. On solidification of the top surface, CO is no longer able to escape and the back pressure thus built up reduces the effective rate of reaction to zero. All gas evolution is stopped and normal columnar crystallization continues (1/*B*). Primary-blowhole formation stops and, as other observers have noted, it generally stops rather suddenly and completely. At this stage central solidification has not yet set in, and there is still a marked tendency for the dendritic crystals to be formed at right angles to the mould surface. The conditions are the same as with killed steel, and the metal is setting solid with no blowholes throughout the whole length of the ingot.

Since there is no sweeping action of the gases the advancing solid wall entraps the mother liquor and there is an increasing concentration of the segregating elements as the solidification proceeds towards the centre (1/*C*). With the slowing down of the ingrowing rate of solidification and the commencement of central solidification, the mother liquor becomes increasingly concentrated in carbon and probably in oxygen, neither of which are being evolved. At the same time the steel is so reduced in temperature that it is becoming mushy. The high concentration of impurities is more pronounced at the ingrowing wall. As compared with fully killed steel, however, there remains a higher free oxygen in the metal, which is only restrained from showing reactions due to the high back reaction rate.

The increasing carbon, and probably oxygen, concentration eventually shows its effect on the reaction by local evolution of CO at the ingrowing wall. With the metal being mushy and the excess-forward rate small, the gas usually forms in the nature of globules which remain *in situ* (2/*C*), and are known as the secondary blowholes.

The same conditions occur in the central part of the ingot, again with the formation of globular blowholes. Here they occur erratically, and form the so-called central blowholes.

The formation of all these types of blowhole tends to force the still-molten liquor upwards, the only direction in which it is free to move. If this upward forcing is sufficiently great, while the top is still partly open, the liquid metal appears to boil through and forms a cauliflower top, when mushy and just on the point of setting as in ingot No. 0. Once the top has set, however, any forcing upwards arising from secondary- and central-blowhole formation can only tend further to eliminate any possibility of a shrinkage cavity. Box-hat steel shows no shrinkage cavity.

Rimming Steel

Coming down the scale of initial free-oxygen content, the next type of steel encountered is the so-called rimming steel. A box-hat steel is a true rimming steel, and in succession the types are called sinking, level, and rising rimmers. With decreasing initial free oxygen, box-hat formation lessens, due to decreasing initial rate of gas evolution, until the metal sets level at the height to which it is poured in the mould. This is known as a level rimmer. Immediately before this stage is reached, box-hat formation is so slight that instead of appearing as a box-hat, the setting ingot shows rim formation at slightly lower levels progressively inwards, thus giving rise to the type known as sinking rimmer.

With decreasing initial oxygen beyond the level-rimming state, the setting ingot rises in the mould to an increasing extent until a stage which is described as fully rising is reached. So far as the range from ingots 0 to 5 is concerned the simplest way to look at the sinking or rising action, perhaps, is that the net result at any initial free-oxygen content is determined by a balance between (1) the action of sinking in wild steel arising from frothing up during teeming, and (2) the action of rising which will be described later for the fully rising steel. In a level-rimming steel the two actions balance : between level-rimming and wild, the sinking action preponderates ; between level-rimming and fully rising, the rising action is the more pronounced.

As the initial free oxygen decreases, stage (2/*B*) is reached at an earlier time in the solidification cycle. This means that primary-blowhole formation starts earlier, and results in a thinner solid skin between the outside of the ingot and the start of the primaries, a fact which is shown in the present series, and is generally well recognized.

Further, the lower initial free oxygen results in stage (2/*B*) being reached earlier for any particular level of the ingot, because of the lower initial difference between the forward and backward rates of reaction. This means that primary-blowhole formation appears at increasing heights in the ingot until finally it occurs in practically the whole length.

In typical level-rimming steel it is found that primary blowholes extend about half-way up the ingot and give a moderate skin thickness. Except for the formation of the box-hat, the setting of the level-

rimming ingot (No. 1) follows the lines already described for a box-hat steel. Ingot No. 2 is a slightly rising rimmer and is a commonly made commercial steel. This type is more usually referred to as a 1-, 2-, or 3-in rising rimmer, indicating the amount of rise from the level to which the steel is teemed in the mould. Ingot No. 3, which is also a rising rimmer, would be considered a very poor example of a commercial-rimming type, as there is excessive rise.

Ingot No. 4 did not show any rimming action and would be classed as a rising steel. Although the external appearance during setting shows none of the characteristics of a rimming steel, the internal structure is very similar. The main differences are the very large rise from the original teeming level, and the very small depth of solid outer skin. When made as a commercial steel, the rise is generally suppressed by capping the ingot mould and allowing the steel to rise only until it solidifies on to the cover. Even when primary-blowhole formation extends over practically the whole length of the ingot, as in No. 4, the rate of oxygen removal is such that the top is sealed over before all the free oxygen has been removed. This implies the subsequent formation of secondary and central blowholes, and of a solid zone between the primaries and the secondaries.

Fully Rising Steel

With further reduction of initial oxygen content, the condition is eventually reached where the primary-blowhole formation ceases, not from closing over of the top as previously, but from exhaustion of the supply of free oxygen to the extent that the carbon–oxygen reaction is balanced. This condition must arise somewhere in the decreasing-oxygen series, and should occur first at the base of the ingot where the back reaction rate is greatest. As a consequence of the balanced reaction the secondary and central blowholes disappear, starting from the base of the ingot and reaching to higher levels as the initial free oxygen decreases. No. 4 ingot shows primary, secondary, and central blowholes throughout the whole length of the ingot, while No. 5 with a lower initial free oxygen shows primaries for the whole length, but secondaries and central blowholes only at the top end.

With a somewhat lower free oxygen than represented by No. 5 ingot, it is suggested there would be primaries extending from the bottom to the top of the ingot, with no secondaries, and either no central blowholes, or only a few, largely at the top as in No. 6 ingot. This type represents what may be called a " fully " rising ingot. Although No. 6 did not rise at all, there is no doubt that with a smaller difference in the deoxidizing addition there would have been a gradually decreasing rise between No. 5 and No. 6.

The conditions during setting of a " fully " rising steel are considered in relation to the evidence of No. 5 ingot. The weight of metal in the mould on completion of teeming is the same as in a killed ingot. After formation of the chilled skin, primary blowholes start to form almost immediately owing to the relatively low rate of reaction ($2/B$). In the range of steels from box-hat to rising rimmer, there is a decreasing amount of reaction of the most rapid type ($2/A$), and thus a decreasing thickness of solid skin below the chilled layer. At a fully rising steel this is non-existent. The initial rate of reaction has fallen so low with decreasing initial free oxygen that only type ($2/B$) occurs after the initial chill-skin formation.

With the formation of these primary blowholes in the solidifying outer zone, the liquid metal is forced higher up in the mould. It seems to rise smoothly and evenly from the moment of completion of teeming. The pushed-up liquid metal chills on the wall of the mould, and the process of chilled-skin formation and start of primaries begins all over again at the higher level. Finally, as the mushy stage is reached the rising molten liquor just cauliflowers in the central top surface zone.

Primary-blowhole formation is stopped by the carbon–oxygen equilibrium being reached before the top is closed over. There is no secondary- or central-blowhole formation, except at the top central position, at which point the known heavy concentration of the mother liquor may be sufficient to restart reaction and form globular blowholes ($2/C$).

Balanced Steel

The next ingot in the series (No. 6) is a semi-killed type, which shows that the change from fully rising takes place over a rather restricted range of free-oxygen decrease. No true " balanced " ingot was obtained. As initial free oxygen decreases, the forward rate of reaction decreases and tends to be balanced at the bottom of the ingot by the relatively higher back rate of reaction due to pressure. Primary blowholes thus disappear upwards from the bottom as initial oxygen decreases. They also decrease in length inwards owing to quicker neutralization of the available free oxygen. There are no secondary or central blowholes, except for a few of the latter at the top central zone. The largest rising ingot from the second list (Figs. 19, 20) is suggested as showing the next stage arising from the lowering of free oxygen from the hypothetical fully rising type.

Due to decreasing primary-blowhole formation there is a decreasing rise at the top of the setting ingot. Finally, all primary blowholes disappear throughout the whole length of the ingot. At the stage where there is no rise but just a slight boiling-through during setting, the type of steel known as balanced is reached. This type contains top central blowholes as in No. 6 ingot, and it may sometimes contain a few primary blowholes at the top end.

Semi-killed Steel

With a very low free-oxygen content, the semi-killed type of steel is formed (No. 6) with no primaries, secondaries, or central holes. However, the finally solidifying highly segregated zone at the top centre becomes sufficiently high in carbon and oxygen for blowhole formation to start in the mushy liquor ($2/C$). Top central holes, as in No. 6 ingot, are formed very late in the setting time, and will often fill up what would otherwise have been the shrinkage cavity of the next type-killed steel.

Killed Steel

With almost zero free oxygen, where the forward reaction rate is balanced by backward reaction rate at atmospheric pressure, and the final heavy segregation at the top centre is not sufficient to restart reaction, a killed steel is formed. There are no primary, secondary, or central blowholes, and a shrinkage cavity is formed. The essential difference between a semi-killed and killed steel would appear to be that the former gives blowholes at the top centre whereas the latter gives a shrinkage cavity.

SEGREGATION OF THE ELEMENTS

Since the present series was produced all from one cast of steel, it is possible to get some comparison between the degree of chemical segregation in different types. When crystallization starts the composition of the initial skeleton crystallites is lower in all elements, except iron, than the liquid from which it is forming. The mother liquor locally must therefore become enriched. If there is no disturbance of this mother liquor, as in killed steels, it is entrapped in the growing crystals, and the mean composition of the area approaches fairly closely that of the steel as teemed. With sufficient gas evolution, however, the mother liquor is removed from the area of its formation, and the further solid dendrites forming are of similar purity. In this case less of the impure mother liquor is entrapped in the advancing wall.

While the points of maximum segregation and to a less extent of maximum purity are of considerable importance, the general level of segregation is usually the first consideration, when comparison is made between ingots. In order to make such a comparison in the present case, the points taken for analysis were intended to represent (1) the rim, and (2) the core, while the known heavily segregated points were avoided. No attempt was made to examine horizontal variations, the general pattern of which has been so thoroughly demonstrated by Swinden,[7] Binnie,[8] and other investigators so far as rimming steel is concerned.

From Figs. 4–7, it is seen that the average rim or outer zone analysis in the killed and semi-killed ingots coincided with the mean analysis of the ingot. In the ingot (No. 5) which gave the maximum rise of the series, the carbon, sulphur, and phosphorus are definitely lower than in killed steels. The rim in the remaining ingots (0 to 4) shows a distinctly lower analysis than the average liquid metal during teeming, but they are reasonably constant among themselves.

While the formation of the purer outer zone would of course be anticipated from earlier work,[7, 8] its steady level of purity from ingots 0 to 4 was not expected. This, however, may be explained by assuming that the solidifying crystallites are of a composition corresponding to the solidus of the steel. The solidifying metal cannot be more pure than this composition, and, once the gas stream has become sufficient to remove all mother liquor, which may be at a low rate of gas evolution, there would be no further drop in the rim analysis with higher rates of evolution. However, the composition of the outer zone for each of the elements shown on the graphs does not agree at all satisfactorily with the position of the solidus usually shown in equilibrium diagrams.

Turning now to the average analysis in the core zone, it is largely complementary to the average analysis in the rim zone. In the case of killed steels there is practically the same analysis in both zones for carbon, sulphur, phosphorus, and manganese. In the rimming steels the steady level of the sulphur and phosphorus below the average liquid metal analysis for ingots 0 to 4 is reflected by a similar constancy above it. Carbon, however, shows a different picture due to loss of carbon as carbon monoxide. Although showing a reasonable agreement between rim and core in the killed steels, and a rise in the core of No. 5 ingot corresponding with the decrease in the rim, there is a marked decrease in the average carbon of the core from No. 4 to 0 ingot, which also affects the average carbon for the whole ingot when solid.

Finally, the well-known rise in carbon, sulphur, and phosphorus from base to top in the core is more marked in the rimming ingots than in the rising, and is least in the killed steels.

Acknowledgments

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The Effect of Shot-Peening upon the Corrosion-Fatigue of a High-Carbon Steel

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SYNOPSIS

Steel peened in seven different ways was subjected to corrosion-fatigue in very dilute sulphuric acid and in sea water ; fine-ground unpeened steel was tested for comparison. It was found that peening greatly increased the endurance at stress ranges commonly prevailing in service. The addition of sodium carbonate and sodium bicarbonate to sodium chloride diminished the endurance of peened specimens, probably by localizing the action ; at high alkali contents, peened specimens have a shorter life than finely ground specimens.

Introduction

Nature of the Problem

It has been known for many centuries that the cold hammering of bronze, iron, and steel improves certain o the intrinsic properties of these metals. The process came to be known as " peening." The name was later applied to other treatments which tend to flatten the surface, such as bombardment with round shot. The word should not be applied to roughening treatment, such as that produced by bombardment with similar shot in the broken condition, or with specially prepared grit.

In recent years, shot-peening has completely replaced hammer-peening. Formed from commercial chilled iron, the shot are usually about 0·05-in dia with well-rounded contours ; they are projected on to the work by, (*a*) air-blasting, in which they are carried by means of a high-velocity air stream through a nozzle, or (*b*) airless-blasting, in which they are propelled by a bladed wheel rotating rapidly, and projecting the shot through an aperture. As each shot strikes the surface of the work, it makes a slight indentation ; a residual stress will be left in the metal around and below every point of impact, with the result that a shot-peened surface is left in a state of strain (the system of stresses is internally self-balancing). The situation is, no doubt, complicated when considered in detail, but some authorities think that the result, considered macroscopically, approximates to a state of fluid compression. The condition of the metal is probably fairly uniform at all parts of the surface and may extend downwards into the metal for about 0·01 in.

It is to be expected that such a treatment would improve the fatigue resistance of many machine and structural parts. From experimental evidence, the tensile half-cycle of an alternating fatigue stress does more fatigue damage than the compressional half-cycle. Consequently, the residual compression in a shot-peened surface should reduce the severity of the tensile half-cycle in the case of any component in which the fatigue stress is a maximum at the skin, such as a rotating beam, or a vibrating bar.

Actual experiments indicate that shot-peening does greatly prolong the life of parts subjected to fatigue. Generalization is difficult, but cases are on record where working lives have been improved up to twenty or more times for particular machine parts.[1, 2] Such a marked improvement cannot be expected, however, where the components are under push-pull stressing. In such a case the stress distribution is uniform over a cross-section, and a fatigue nucleus could appear anywhere in the whole volume of stressed metal, and not necessarily close to the surface. Skin imperfections certainly exist, and perhaps exert weakening influences slightly exceeding those due to chance internal imperfections. It follows that shot-peening may be expected to improve only slightly the fatigue properties of highly polished components under push-pull stresses ; where components are left with rough surfaces, peening should improve the endurance to fatigue of this kind to a greater extent.

In the case of corrosion-fatigue, destruction always starts from the surface of a stressed part. Attack by the conjoint chemical and mechanical action causes narrow fissures, which grow inwards from the surface of the metal until they become cracks. From their nature, the residual compressions left by shot-peening must surely delay the inception of corrosion-fatigue fissures, and perhaps even for a time their extension after their initiation ; firstly, by diminishing the virtual effect of the tensile half-cycle of fatigue stressing, and secondly, by closing any incipient surface cracks, which would otherwise constitute stress-raisers, and also provide places favourable to corrosion.

Consequently, it is to be expected that shot-peening will improve the corrosion-fatigue resistance of any metal component, irrespective of the kind of fatigue stress which operates.

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The Plan of the Present Research

The research was undertaken to investigate the corrosion-fatigue properties of peened test-bars manufactured from a single cast of steel. Some bars were kept in the "as-ground" condition for testing as controls ; the rest were peened. Seven kinds of peening treatment were employed.

Experiments were conducted with the following corrosive liquids :

(*a*) Aerated 0·005 normal H_2SO_4, a concentration which was thought to exceed the acidity of the condensate (dew) deposited in the evening on steel surfaces exposed to industrial atmospheres
(*b*) Aerated sea water
(*c*) Aerated 0·5 normal NaCl containing different concentrations of an equimolecular mixture of sodium carbonate and sodium bicarbonate.

Specimens were tested under reversed stress on a rotating machine fed with corrosive liquid, until fracture occurred. The data were expressed as stress-endurance (S–N) curves, which provide a rapid assessment of the effects of different kinds of peening.

EXPERIMENTAL SECTION

The steel used was supplied by Rylands Bros., Ltd., Warrington, through the kindness of Mr. R. S. Brown, and had the following chemical composition :

C, %	Si, %	S, %	P, %	Mn, %
0·80	0·17	0·039	0·024	0·57

The steel was rolled to 2-in billets, reheated, and again rolled, then pickled in HCl, limed, and drawn unpatented to 0·5 in. The rods were patented by heating to 1030° C, and cooled through lead at 502° C, the process being continuous. The stock was then drawn, in two passes on a rack bench, to the required size of 0·375 in. It was straightened and polished by reeling. During reeling the diameter increased by 0·004 in, and each length was ground to remove this excess. Finally the material was cut into test-bars 12 in long ; these were finished by centreless grinding to 0·360 ± 0·001-in dia. The final operation produced an exceedingly good surface finish.

One third of the ground bars were preserved for testing as controls. The rest were peened, according to standard practice, by Messrs. J. W. Jackman and Co., Manchester, through arrangements made by Mr. W. E. Ballard and Mr. F. W. Neville. Only the central three inches of each bar received peening.

The conditions of peening were :

Nozzle :	2·5 in × 0·125 in, slightly divergent to cover a length of 3 in at a distance of 6 in from the nozzle tip
Time of treatment :	5·2 sec per bar
Air pressure :	As stated below
Shot sizes :	*Large*—passing mesh $\frac{1}{21}$-in square opening, not passing mesh $\frac{1}{38}$-in square opening
	Small—passing mesh $\frac{1}{38}$-in square opening, not passing mesh $\frac{1}{50}$-in square opening
Grit sizes :	*Large*—commercial grade D
	Small—commercial grade E

Seven types of peening were used, shown as follows :

Group	Air Pressure, lb/sq.in.	Type of Peening Treatment
1	30	With large shot
2A	30	With large shot, followed by treatment with small grit for 5·2 sec.
3	30	With small shot
4	40	" " "
5	50	" " "
6	30	" large grit
7	30	" small grit

In Group 2A it was hoped to produce a surface retaining the advantages of peening but sufficiently rough to provide good adhesion when a sprayed metal coating was subsequently applied.

The chemical solutions were made up from B.P. reagents and distilled water. The sea water, taken from Chichester Harbour, was supplied by Dr. Slater.

Tensile tests were conducted upon bars of unpeened material, 0·20-in dia, and furnished the following mean values :

Ultimate tensile stress, tons/sq.in.	90·5
Elongation on 2 in, %	5
Reduction in area, %	36

No yield could be detected during the tests.

Apparatus

The machines used for corrosion-fatigue testing were of the rotating-beam type. They were similar to those used previously in the laboratory by Huddle,[3] modified in detail to reduce torsional oscillation. Four-point loading was provided through four ball

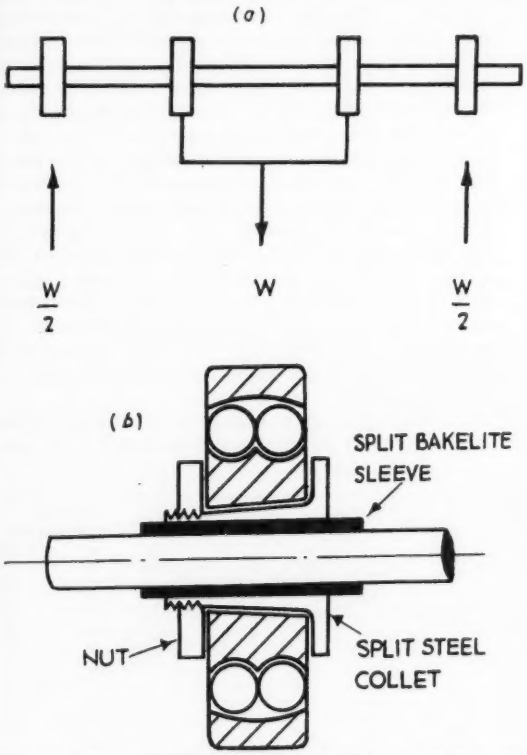

Fig. 1—(*a*) Arrangement for four-point loading in fatigue-testing machine; (*b*) Method of securing ball bearing

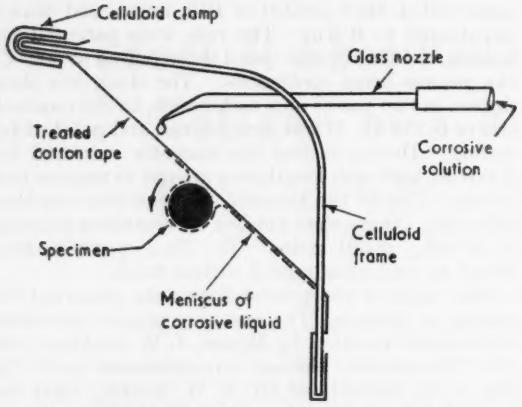

Fig. 2—Arrangement for applying corrosive liquid

bearings which firmly engaged the specimen by means of inserted split-collets and split bakelite sleeves, as shown in Fig. 1.

The test-bars were of parallel section, 12-in long and 0·360-in dia. Before a test, the middle portion of the specimen was carefully degreased by means of repeated swabbings, first with benzene, then with carbon tetrachloride, and finally with acetone.

The corrosive solutions, previously shaken with air, were carried in a reservoir above the apparatus, and were allowed to drip upon a cotton-tape sling[3, 4] which served the corrosive to the specimen as shown in Fig. 2.

Method

Specimens were placed in the machine and then rotated at 1500 r.p.m. The load was then applied, the corrosive solution served, and the revolution counter started. Each test was continued until the specimen broke, when the machine was stopped automatically. The counter reading gave the endurance of the specimen at the particular stress employed.

Results

The main results were expressed in the form of stress-endurance curves:

(*a*) Showing the fatigue properties of uncorroded and unpeened steel
(*b*) Comparing the behaviour of the steel in various peened and unpeened surface conditions, when subjected to corrosion-fatigue in dilute acid
(*c*) Comparing the behaviour when subjected to corrosion-fatigue in sea water.

In those experiments designed to test the effect of adding sodium carbonate and sodium bicarbonate, curves were plotted showing the relation of the endurance numbers to the concentration of the added salts.

The alkaline buffer solution (X.M. Na_2CO_3 + X.M. $NaHCO_3$) was used at a number of different concentrations varying from X = 0 to X = 0·7; the upper limit of X was imposed by the limited solubility of $NaHCO_3$.

The air-fatigue properties of the steel, very finely ground with a sequence of emery papers ending with 0000 and then covered with unmedicated vaseline, are indicated in Figs. 3 and 4. Recorded there also are the results obtained in 0·005 N sulphuric acid and sea water respectively. Control specimens, with the surface "as-ground" (not peened), are included in the figures.

Figures 3 and 4 may be considered together. They indicate that peening causes a marked improvement of endurance at reasonable working stresses, under the specific conditions of testing used. Some types of peening were found to be very much better than others.

The results obtained in M/2 NaCl made alkaline with sodium carbonate/sodium bicarbonate buffer are shown in Fig. 5. It will be observed that whatever the concentration of the buffer, the peened bars always had a greater endurance than unpeened bars in the unbuffered solution. On the other hand, at high alkali concentrations, the peened specimens had shorter lives than unpeened specimens exposed to the same solution; thus peening does not always bring about an improvement.

DISCUSSION

Preliminary Remarks

Before a critical consideration of the results is attempted, the following remarks are offered:

(*a*) The steel adopted for this work had an unusually large carbon content, approximating to that of a spring steel. It could not be regarded as typical of steel used for general engineering purposes. When the work was started, it was the only metal of high quality readily available for the production of the specimens.

(*b*) The test-bars were ground to a parallel section without any reduction of diameter in the middle: it was the only machining possible at the time. In consequence, heavy stress-loading of the test-bars caused severe frettage-corrosion at the sleeves of the middle bearings in the machines. Breakage often occurred at these points, so that values of the endurance at high stresses were difficult to obtain. By the use of bakelite sleeves and a lubricant of zinc oxide in vaseline, some values were secured, but only after a considerable wastage of specimens. The difficulty was more pronounced with sea water than with weak acid, because of lateral creepage.

(*c*) The experiments had to be conducted in an open laboratory because a thermostated room was

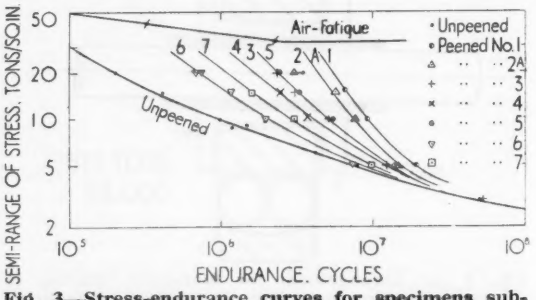

Fig. 3—Stress-endurance curves for specimens subjected to corrosion fatigue in dilute sulphuric acid

not available. However, thermometer readings were recorded and averaged during every experiment, and a temperature correction was applied to each endurance so that standardized values could be plotted. The factor used, namely a 3·3% increase of endurance per degree F fall of temperature, was obtained from previous work.[5] The corrected results fell on curves in a satisfactory manner, whereas the uncorrected ones were badly scattered.

(*d*) The peening was conducted upon test-bars which had been accurately machined and highly finished. In practice, this procedure would be considered unnecessary and expensive, since a rough finishing could be used without disadvantage, but it is not absolutely certain that the same benefits would be obtained from peening a rough surface.

Interpretation of Results

That a peened surface suffers less damage under corrosion-fatigue conditions than a fine-ground surface in acid or neutral solution, is not surprising. Damage due to dry fatigue is diminished by peening, probably for two reasons; the process closes up any submicroscopic fissures which would form the most favourable starting-places of fatigue cracks, and the residual stresses, being additive to the applied stress, tend to diminish the resultant stress during the dangerous tensional half-cycle, although increasing it on the less dangerous compressional half-cycle. Thus it can be understood that the transformation of the saucer-shaped cavities, characteristic of stressless corrosion, into the cracks characteristic of corrosion fatigue—a development studied by McAdam and Geil,[6] and later at Cambridge by Simnad[7]—will occur less readily and less rapidly on a peened, than on an unpeened, surface.

Up to a point, the improvement might be expected to increase with the momentum of the shot-particles. The curves corresponding to steel peened with small shot at 30, 40, and 50 lb/sq.in. respectively show that the lives increase with the momentum in sea water (Fig. 4), but that in acid (Fig. 3), the steel specimens peened at 40 lb/sq.in. have consistently shorter lives than those peened at 30 and 50 lb/sq.in.; this is not easy to explain.

Another strange result is that in solutions of sodium chloride containing relatively large amounts of alkali (Fig. 5), peened steels have shorter lives than unpeened. This is of great practical importance, since

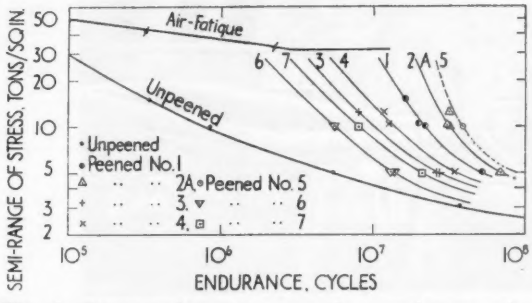

Fig. 4—Stress-endurance curves for specimens subjected to corrosion-fatigue in sea water

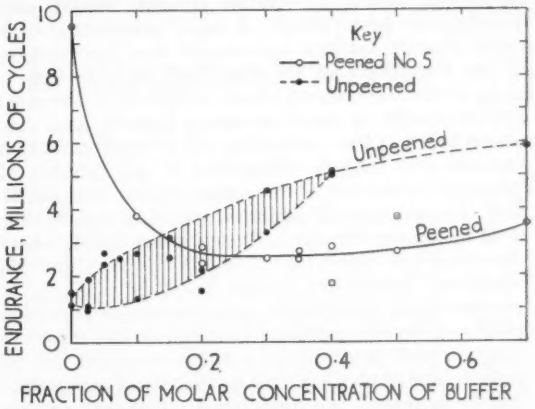

Fig. 5—Endurance curves (at ± 10 tons/sq.in.) for specimens subjected to corrosion-fatigue in sodium carbonate/sodium bicarbonate buffer solution of different concentrations

saline waters which have been treated with alkali with a view to preventing stressless corrosion must be regarded as likely to produce corrosion-fatigue more readily on a peened than on an unpeened surface.

An explanation may be tentatively suggested. The addition of alkali in small amounts to sodium chloride is known to diminish the corroded *area* more quickly than it diminishes the *total attack*—for electrochemical reasons discussed elsewhere[8]—thus increasing the intensity of corrosion; larger amounts, however, may stop corrosion altogether. In the absence of stress, waters containing an insufficient amount of an alkaline inhibitor produce pitting and rapid penetration into the steel, so that the treated water will perforate a plate more rapidly than the same water free from inhibitor, although the total destruction of metal is less. Under conditions of corrosion-fatigue, where the attack takes the form of a narrow fissure rather than a hemispherical pit, it might well be expected that small amounts of alkali would increase the rate of penetration of the crack, whereas larger amounts would decrease it, especially since, as shown by Simnad,[7] a lone crack produces far more weakening than a family of cracks close together. The curve corresponding to a peened surface (Fig. 5) shows the decrease of endurance produced by small additions, and suggests, less decisively, that there is a perceptible recovery at high concentrations.

Finely ground (unpeened) specimens show a distinct increase of endurance when alkali is added, although reproducibility here is poor. The behaviour of the specimens is probably connected with the fact that grinding leaves a very complicated system of internal stresses, which are in places favourable to penetration and in other places unfavourable. In a water containing an inhibitor, corrosion-fatigue consists of two stages; a period which elapses before the breakdown of the surface film, and a period needed for the advance of the crack into the interior. In acid, and even in neutral solutions, the first period is short, but the authors have shown[9] that in neutral solutions it is greatly lengthened by the presence of a chromate inhibitor; probably an alkaline inhibitor (sodium

carbonate) added to sodium chloride would also lengthen the first period. A rapid penetration of a crack into the interior will require that the position of the breakdown of the film shall coincide with a place where the internal stress conditions in the steel are favourable to rapid extension inwards as a corrosion crack. The probability of a rapid extension inwards will thus be diminished if the number of points of breakdown of the surface skin is diminished, and the addition of inhibitor will accomplish this, thus increasing the life. On the other hand, where the metal just below the surface is in a very uniform condition, breakdown at one point is as likely to produce rapid penetration as breakdown at any other point. If by the addition of inhibitor the attack is concentrated on a limited number of points instead of being distributed over many, the rate of penetration will be increased, not diminished. This may explain why the lives of peened specimens fall below those of unpeened specimens when the concentration of alkaline buffer is sufficient.

Acknowledgments

In addition to the acknowledgments already made of the services rendered by Mr. W. E. Ballard of Metallisation, Ltd., Mr. R. S. Brown of Messrs. Rylands Bros., Ltd., and Mr. F. W. Neville of Messrs. J. W. Jackman and Co., the authors thank Mr. C. A. Taylor and Mr. Clifford Evans for their assistance in the experimental work. They also acknowledge the financial support of the British Iron and Steel Research Association and assistance from members of the staff of that organization.

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Pouring Liquid Steel—from original woodcut by Viva Talbot

Ingot Surface Defects in Structural Steels

By L. Reeve, Ph.D., B.Sc., F.I.M.

SYNOPSIS

The causes of ingot surface defects in structural steels are considered, the controlling factors concerned being divided into two main groups, namely (1) those acting in the furnace and largely associated with steelmaking chemistry and (2) those acting outside the furnace and associated with temperature and teeming conditions. The chemical factors considered are sulphur, carbon, manganese, oxygen, and hydrogen ; the temperature and teeming conditions are discussed from the point of view of their influence on the rate of growth of the ingot skin during and immediately after teeming. The general discussion is illustrated by the results of a detailed statistical analysis of the causes of defects in steel produced from 19,600 tons of 85-cwt. ingots of " balanced " quality carbon steel used for the production of steel joists, and it is shown that the lowest steel defectives are obtained in the lower-sulphur groups and in steel at an intermediate temperature and that smaller nozzles are preferable, particularly with hot steel and high sulphur. Defects produced by sub-surface blow-holes in balanced quality steels are also dealt with, and a description is given of a technique for the cutting of corner samples from large ingots of the type of steel concerned ; some of the factors controlling skin depth in the ingots are discussed.

Introduction

Over a long period of years practical steelmakers have learnt much regarding the causes and effects of ingot defects. Nevertheless, examination of the records of British and American open-hearth conferences reveals that there is still far from complete agreement about many of the factors involved in the production of defective steel. The reason of course lies in the large number of controlling variables which may be acting simultaneously during the melting, teeming, and processing of a cast of steel, and the mutual and possibly contradictory effects of these variables are difficult to disentangle with any degree of accuracy. In the present paper an attempt has been made to meet this difficulty by presenting conclusions based upon a considerable body of detailed steelmaking records, and statistical analysis has been applied to the main results to confirm their significance.

The paper deals mainly with structural mild-carbon steels as used for plates and sections in the tensile range 26–33 tons/sq. in., though some of the general conclusions apply to alloy steels. Most of the steel used for structural purposes is of the so-called " balanced " variety—sometimes called semi-killed—in which a very weak rimming action is controlled by deoxidizers to an extent sufficient to give a thin-skinned ingot with sub-surface blow-holes, without undue segregation. The production and influence of these sub-surface blow-holes will be discussed in some detail.

GENERAL CAUSES OF STEEL DEFECTS

It is generally agreed that there are two main groups of controlling factors in the production of ingot defects, namely (1) those acting in the furnace and largely associated with steelmaking chemistry, *e.g.*, sulphur, slag conditions, oxygen and hydrogen contents, and similar chemical factors, and (2) those associated with teeming conditions, *e.g.*, (i) rate of teeming, (ii) temperature of the steel (this is largely determined in the furnace but produces most of its effects during teeming), (iii) mechanical features such as splash and surge, and (iv) influence of mould design.

The author is aware that many steelmakers believe that certain defects are produced in perfectly sound steel by the rolling mills. Such defects are outside the scope of this paper, since they are not essentially ingot defects, but certain types of defects produced in soakers and reheaters will be mentioned.

FACTORS ASSOCIATED WITH STEELMAKING CHEMISTRY

Influence of Sulphur

In considering the influence of steelmaking chemistry in detail, the influence of sulphur should be mentioned first. The influence of sulphur on the hot strength of steel is well known and explains its almost universally agreed bad effect in ingot cracking. The work of Hall[1, 2] for the Steel Castings Research Committee,* in which he measured the strength and ductility of different types of steel at very high temperatures, is worth referring to in this connection.

Hall showed that 0·07% of sulphur in a 0·12% carbon steel reduces ductility to zero down to temperatures of 1280° C., whereas with 0·02% of sulphur and 0·10% of carbon the ductility was about 55% at the temperature mentioned. All Hall's steels were silicon-killed, and his results cannot be transferred bodily to semi-killed or balanced steels. The influence of silicon range from a trace to about 0·3% in high-temperature cracking needs further investigation but is complicated by the difficulty of avoiding blow-hole formation in the low-silicon steels.

This is a paper of the Ingots Surface Defects Sub-Committee of the Steelmaking Division of the British Iron and Steel Research Association, received 12th May, 1948. The views expressed are the author's and are not necessarily endorsed by the Sub-Committee as a whole.

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* A joint committee of The Iron and Steel Institute and the British Iron and Steel Federation, then working under the auspices of the Iron and Steel Industrial Research Council.

Influence of Carbon

In practice it is known that the deleterious influence of sulphur on the cracking of structural-steel ingots is most marked in the carbon range 0·15–0·20% and is much less marked in lower-carbon steels. This ties up with results reported by the author at an open-hearth conference in 1943 and is illustrated in Fig. 1, which shows a maximum defective range in structural steels lying between 0·15% and 0·20% of carbon. It was then suggested by Binnie that the optimum cracking effect at about 0·16% of carbon was due to the fact that the volume change at the peritectic temperature (1492° C.) is at a maximum at or about this carbon content.

Of more direct bearing upon the problem might be the experimental observation of Hall[1] which indicated that the range of temperature below the solidus in which steels have less than 5% ductility is widest at about 0·20% of carbon and is very narrow at 0·10% of carbon. In the latter case steel has a 5% ductility at temperatures above 1400° C., whereas at 0·20% of carbon this ductility is not attained until the steel has cooled to about 1280° C.

Influence of Manganese

Hall[1] showed that increasing the manganese content to 1·5 and 2·0% improved high-temperature ductility. The combined effects of manganese and sulphur were not studied ; such a study might have proved instructive, particularly as regards the cracking of free-cutting-steel ingots.

The influence of manganese is complicated by its combined effect as a sulphide-former and deoxidizer, and by the further complication that the equilibrium $MnS + Fe \rightleftarrows FeS + Mn$ moves to the right at high temperatures. Rapid cooling of the ingot skin tends to freeze the equilibrium on the FeS side, despite the presence of more than sufficient manganese to form MnS. An increase of manganese should move the reaction more to the left. These factors no doubt have some influence on the production of ingot cracks, and the presence of FeS in the neighbourhood of such cracks has been reported. On the other hand, the presence of FeO and MnO, and the formation of oxide–sulphide complexes, may modify the influence of FeS, as mentioned later.

Influence of Oxygen

Lower-carbon steels are usually higher-oxygen steels, and it is tempting at this stage to embark upon a discussion of the possible influence of oxygen on ingot defects. Much of the information on this problem is still not clear and it is not proposed to say anything more on this subject in the present paper, apart from recalling the generally accepted opinion that an optimum oxygen content, whether measured by iron in the slag or otherwise, is necessary for each class of steel. In particular, the thickness of ingot skins in rimming and balanced steels is almost certainly controlled by the oxygen content of the steel. This point will be referred to again later, and at least one type of steel defect which is related to skin thickness, and therefore indirectly to oxygen content, will be indicated.

There is a general impression that killed-steel ingots are more prone to cracking than balanced or rimming steels. In this connection attention is drawn to a previous paper by the present author[3] in which it was shown that cracking due to high sulphur in weld metal was least marked in the more oxidized welds. Killing, in the sense of conversion of FeO and MnO to SiO_2, increased sensitivity to sulphur, and there was some evidence that this effect was related to the formation of oxide–sulphur complexes in the more oxidized steels. Although the speed of freezing of weld metal is very much greater than in normal steel-making practice, some of the fundamental cracking factors would appear to apply to both types of steel.

Influence of Hydrogen

The possible influence of hydrogen in steel on ingot defects has often been mentioned, but the subject is even more controversial than in the case of oxygen. The influence of hydrogen on hair-line-crack formation is outside the scope of the present paper, since this type of defect does not normally arise as a serious problem in the type of structural steels which are under consideration. Hydrogen has been mentioned as a possible factor in blow-hole formation in both effervescing and balanced steels. The present trend of opinion, however, is to regard it as secondary to the influence of oxygen (or rather of carbon monoxide), which is regarded as the original source of the blow-holes. It is interesting to mention that it has been reported that gas driven off by drilling into these blow-holes when cold is mainly hydrogen, and this has given some support to the theories still heard occasionally that hydrogen is a factor in blow-hole formation in effervescing steels ; the long controversy over this point is reported in great detail by Hultgren and Phragmén.[4] In the case of killed structural steels there are occasional cases of sub-surface pinhole porosity, which may be due to hydrogen, though little concrete information about this is available.

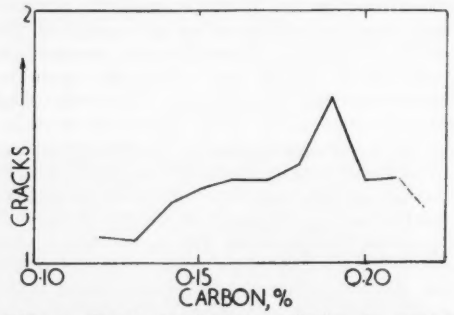

Fig. 1—Crack grading in relation to carbon content (manganese range 0·50–0·60%)

Influence of Rate of Teeming

To turn to the influence of factors outside the furnace on ingot defects, it is fairly generally agreed that the influence of rate of teeming, and with it part of the influence of up-running as compared with downhill teeming, is due to the effect of rate of growth of the initial ingot skin, particularly in the first few minutes. Several investigations[5, 6] have confirmed that the

general law of the rate of growth with time is covered by an equation of the type $d = KT^{\frac{1}{2}}$, where d = depth of solidification from the mould wall (inches), T = time after commencement of teeming (minutes), and K is a constant ranging between 0·87 and 1·37, depending upon the type and size of mould, the temperature of the steel, and similar factors. For approximate calculations K can be regarded as being equal to unity.

To put the matter into practical terms, the first inch of skin is produced in approximately 1 min., whilst it takes nearly 4 min. to produce a 2-in. skin. Hence if a 6-ft. high ingot is rapidly teemed in, say, 1 min., a skin of solid steel 1 in. thick is being relied upon to support a ferrostatic head of 2 atm. It is probable that separation from the mould wall takes place within that time, and as a result the skin may crack or burst, especially as steel is weak and brittle in the temperature range concerned.

Slower teeming and up-running permit the possibility of a thicker skin being formed before the mould is full; moreover, the outer part of this skin will have a greater chance of cooling to a lower temperature, where hot steel has good ductility.

Influence of Teeming Temperature

The equation mentioned for the growth of the ingot skin does not take into account the influence of teeming temperature, though Lightfoot,[6] in his mathematical analysis of the problem, did take into account the temperature of the mould; an increase in the latter slowed down the rate of freezing. Further investigation to cover the influence of steel temperature would be of value, but it is probable that high temperatures lead to a delay in growth of the skin, *i.e.*, they produce a thinner skin by the time the mould is full; hence the known bad effect of high steel temperatures on ingot cracking, which can be counteracted to some extent by slower teeming. High teeming temperatures also have an effect on gas liberation in steels of the effervescing or rimming type, tending to retard it and to produce thin skins with shallow sub-surface blow-holes. The influence of such blow-holes on the rate of solidification of the ingot has not been investigated, but possibly some delaying action will occur.

Low teeming temperatures, whilst preventing ingot cracks, may lead to other troubles associated with teeming conditions. The freezing of splashes of steel to the side of the ingot, surface lapping, double skins, and similar difficulties are more liable to occur with cold steel, which may lead to shelly defects in the final product.

DETAILED INVESTIGATION OF CAUSES OF INGOT DEFECTS

Many of the points which have been mentioned are well illustrated by the results of a statistical examination, carried out in 1943 at one of the works with which the author is associated, of the steelmaking data concerning 19,600 tons of steel used for structural joists in the size range 4 × 2 in. to 6 × 5 in. and weighing 11–25 lb. per foot. The carbon and manganese contents of this steel were reasonably constant in the range 0·18–0·22% of carbon and 0·50–0·65% of manganese. Phosphorus within the normal range of 0·03–0·06% has not been found to be a serious factor in the production of the type of defects associated with these joists, and was not considered in the analysis. The steel concerned was deoxidized with ferromanganese in the ladle, no silicon being used; the steel was of the balanced or "thin skin" rimming type. Aluminium was added to the ingot top only, the normal addition being about 4 oz./ton. The ingots were almost entirely of the 85-cwt., wide-end-down, top-teemed type and were of approximately rectangular section (19 × 27 in.). The ladles were all of one size (65 tons capacity).

The records for the 19,600 tons of steel concerned were collected on suitable summary cards and were divided into sulphur, skull, and nozzle groups. The furnaces concerned are of the tilting type, and normally no slag is present in the ladle. As a result, skull formation is sensitive to tapping temperature, as shown by the figures indicating the relationship between these factors. The skulls are reported by their height as courses (C) of brickwork from 1 course to 10 courses; P represents a plate skull covering the ladle bottom; and PP represents a part plate, *i.e.*, the centre of the ladle bottom is free from frozen steel. With the hottest steel (above 1590° C.) no skull whatever is produced. The influence of pit sulphur and of teeming temperatures (as measured by skull) on defectives in the mild-steel joists are indicated by the following data:

Influence of Pit Sulphurs (all skulls, excluding 6C–10C)

Sulphur Group	Number of Ingots	Defective Index
0·030–0·039	857	10·3
0·040–0·049	1866	10·6
0·050–0·060	953	14·1
Over 0·060	430	17·1

Influence of Teeming Temperature (all sulphur groups)

1½-in. Nozzles

Skull Group	Number of Ingots	Defective Index
Nil + *PP*	973	12·6
P + 1*C*	673	7·7
2*C*–5*C*	836	6·3
6*C*–10*C*	566	20·4

2-in. Nozzles

Skull Group	Number of Ingots	Defective Index
Nil + *PP*	596	21·1
P + 1*C*	589	10·6
2*C*–5*C*	439	17·7
6*C*–10*C*	300	16·4

The relationship between ladle skull and tapping temperature (measured by platinum/rhodium immersion thermocouples) was as follows:

Skulls	Tapping Temperature, °C 1½-in. Nozzles	2-in. Nozzles
Nil	1595	1590
PP	1589	1582
P	1584	1577
1 course	1580	1572
2 courses	1577	1568
3 courses	1574	1564
4 courses	1572	1561
5 courses	1569	1558
6 courses	1568	1556
7 courses	1567	1554
8 courses	1565	1552
9 courses	1564	1551
10 courses	1563	1550

The quantity of defective steel is shown in Figs. 2 to 4 by a "defective index" which represents the

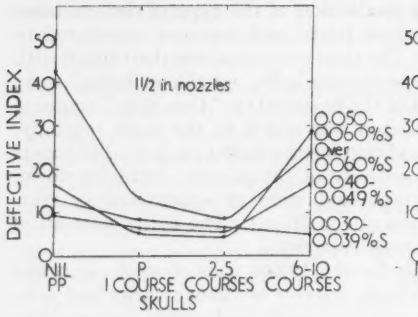

Fig. 2—Defectives in relation to sulphur and temperature ; 1½-in. nozzles. The sulphur ranges are denoted by figures on the curves

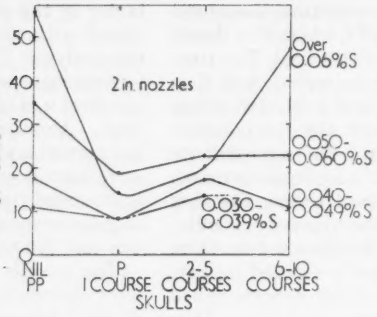

Fig. 3—Defectives in relation to sulphur and temperature ; 2-in. nozzles. The sulphur ranges are denoted by figures on the curves

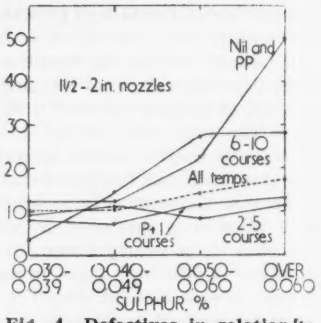

Fig. 4—Defectives in relation to sulphur and temperature ; 1½-in. and 2-in. nozzles. The extent of skulling is indicated by the data on the solid curves; the broken curve represents all temperatures

percentage of the charged weight rejected for various defects (mainly cracks) multiplied by an arbitrary factor. One complication which arose in this analysis was the influence of the difference in weight per foot of the sections concerned which, as already mentioned, ranged from 11 to 25 lb. per foot. Earlier investigations had shown that defectives in joists as measured by percentage loss in weight were, on the whole, proportionate to the weight per foot, *i.e.*, a 25-lb. section could be expected to give twice the percentage defectives given by a 12·5-lb. section. This was known to be true for a range of sizes much wider than those covered by this investigation ; the results for joists up to 41 lb. per foot are shown in Fig. 5. Hence to allow for the different weights the defectives for all the sections under investigation were corrected to a 20 lb. per foot basis, *i.e.*, the defectives in the case of the 11-lb. sections have been multiplied by 20/11, whilst defectives in the 25-lb. sections have been multiplied by the factor 20/25. This procedure is a logical one, since the influence of weight per foot on defectives is too appreciable to be neglected, and the presence of a large number of heavy sections in one particular sulphur or skull grouping would overweight the defectives in that group, unless corrected in the manner described.

Broadly speaking, the main conclusions to be drawn from the results are as follows :

(1) Under normal conditions the lowest defectives are obtained in the low-sulphur groups.

(2) The best overall results are obtained in the intermediate skull groups, both very hot and very cold steel being undesirable.

(3) The 2-in. nozzles give appreciably worse results than the 1½-in. nozzles, particularly with hot steel and high sulphur. With cold steel, however, (6*C*–10*C* skull), this analysis indicates that 2-in. nozzles give better results than 1½-in., except in the highest sulphur group (*see* Figs. 2 and 3).

The position regarding very cold steel (6*C*–10*C* skull) is complicated to some extent by the fact that such

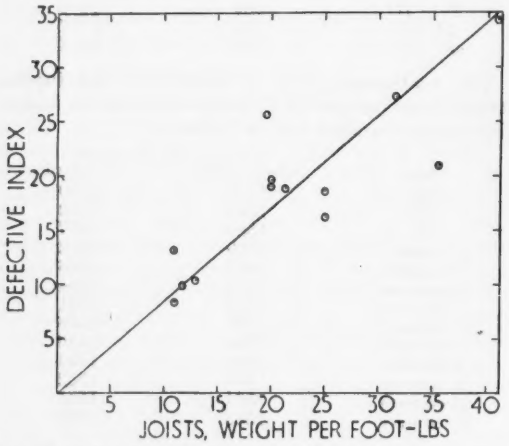

Fig. 5—Relation between weight per foot and defective index for steel joists

Fig. 6—Typical cracks in web and flange of steel channel

Fig. 7—Photograph of red-hot ingot, showing typical ingot cracks

steel gives rise to another type of defect, *viz.*, "shelliness," which may be confused in certain cases with "crack."

As already pointed out, most of the defects concerned in this analysis were cracks, typical samples being shown in Fig. 6. These in turn are known to be produced from transverse ingot cracks of the type shown in Fig. 7. The cracked ingot reproduced in Fig. 7 is of interest as it was taken by "its own light" and shows the cracks as they appear at the approach to the cogging mill after the fourth pass; the cracks are brighter than the main body of the ingot owing to these conditions.

Since the first writing of this report a qualified statistician (Mr. Denis Ward, B.A.) has joined the staff of the Appleby-Frodingham Metallurgical Department. He has re-examined the data presented above and has contributed the following notes to the report.

From the data provided an analysis of variance has been carried out and has shown that at a significance level of 5% :

(1) Increasing sulphur gave increasing crack defectives.

(2) Moderate skulls gave the smallest crack defectives.

(3) This last effect was most marked for high sulphurs.

(4) The 2-in. nozzles gave, on the whole, more defectives than the $1\frac{1}{2}$-in. nozzles, but there was not sufficient information to show the effect of nozzle size in particular sulphur or temperature groups.

The analysis confirms the conclusions regarding defectives shown in the data given on page 172.

The standard error of the defective index for these particular joists cannot be assigned without extensive computation, but in a previous investigation of the defectives in 9000 tons of $5 \times 4\frac{1}{2}$-in. joists, which investigation led to essentially similar conclusions, it was found that:

$$\text{Standard error of defective index} = \frac{126}{\sqrt{n}},$$

where n is the number of ingots.

The number of ingots concerned in the investigation is shown on page 171. Thus, even with 1000 ingots, the standard error of the defective index is about 4.

The need for a considerable body of steelmaking data before significant conclusions can be drawn is therefore emphasized.

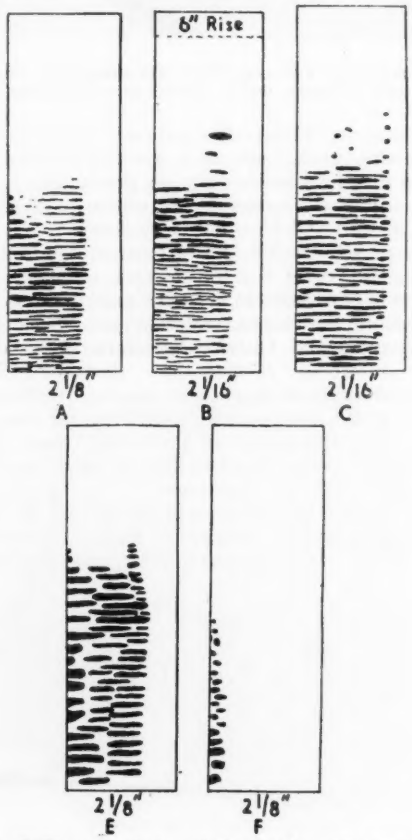

Fig. 8—Influence of teeming speed and up-running on skin depth (Jackson)

SUB-SURFACE DEFECTS

So far, the present paper has been concerned with obvious surface defects in ingots, such as cracks, leading to equally obvious surface defects in the finished product. Some of the more troublesome defects in finished steel are associated with sub-surface blow-holes in the ingot, when such blow-holes are too near the surface and become exposed in the soakers or reheaters.

The study of blow-hole formation in ingots—particularly in ingots of the structural or balanced types of steel has not, in the author's view, received the attention the complexity of the problem deserves, particularly since such steel probably represents the major tonnage of commercial steel made and the

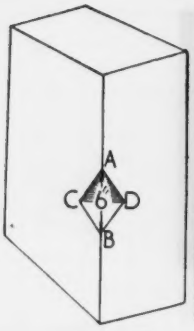

Fig. 9 — Details of corner-sampling method

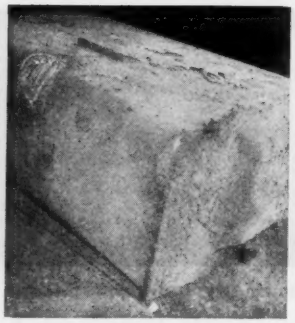

Fig. 10—Ingot corner sampling. General view of 10-ton ingot

Fig. 11—Ingot corner cross-section, showing thick skin. One-half actual size

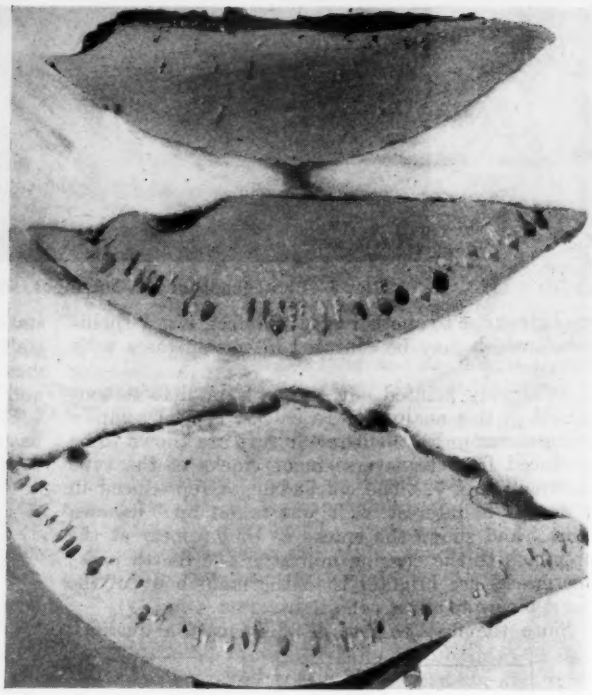

Fig. 13—Ingot corner cross-sections, showing top, middle, and bottom of same ingot. One-half actual size

consistent production of a truly balanced ingot is by no means an easy matter.

The classic work on the subject is that of Brinell and Wahlberg,[7, 8] who were amongst the first to make a study of the influence of degree of deoxidation on type of blow-hole formation. The subject was also studied by the Heterogeneity of Steel Ingots Committee, and similar information was contained in the Fifth Report of that Committee, published in 1933.* The Committee also defined the terms "killed," "semi-killed," "rimming," and "balanced" steels. The Report referred to also included the first of a series of papers by Principal C. A. Edwards and co-workers on the influence of various factors on the position of blow-holes in small induction-furnace ingots of varying carbon and oxygen contents.

Hultgren and Phragmén, in their well-known paper on rimming steels,[4] discuss in great detail the equilibrium reactions involved in gas production in these ingots and pay considerable attention to the mechanism of skin and blow-hole formation.

Two papers by Mr. A. Jackson of the Appleby-Frodingham Steel Co. on factors influencing the position of blow-holes in balanced and rimming steels, read before the Swansea Metallurgical Society and the Newport and District Metallurgical Society in 1936,[9] are worthy of attention. Jackson's method of examination of ingot skins involved drilling into the side of the ingot to a depth of 2 in. and examining the wall of the cavity so produced. Some of his diagrams are reproduced in Fig. 8, where ingots *A*, *B*, and *C* show the influence of teeming rate, and ingots *E* and *F* the influence of top-pouring as against up-running, on the thickness of the skin of rimming-steel ingots. Slower teeming and up-running results

* Fifth Report on the Heterogeneity of Steel Ingots, *The Iron and Steel Institute*, 1933, *Special Report No.* 4.

Fig. 12—Ingot corner cross-section, showing thin skin. One-half actual size

Fig. 14—Ingot corner cross-section, showing thin skin and coarse blow-holes. One-half actual size

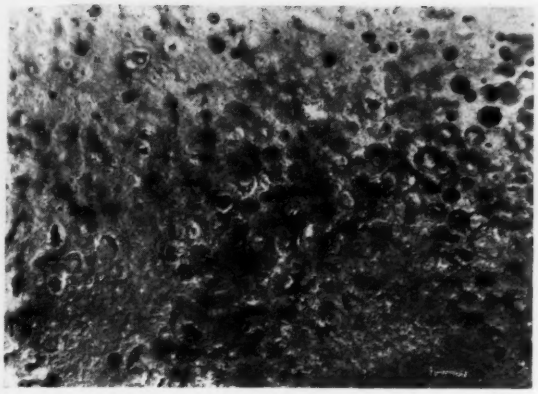

Fig. 15—Exposed blow-holes on specimen shown in Fig. 14, after passage through the soakers

in a thicker skin. Time of filling the moulds varied from 600 sec. for ingot *A* to 90 sec. for ingot *C*. In the original diagrams the full width represented two inches of the ingot wall, whilst the height was drawn on a scale of 1 in. = 1 ft. The diagrams as reproduced in Fig. 8 have been reduced to approximately one-third of the original.

The writer has recently modified this technique by gas cutting from the corner of the ingot under examination a piece of from 6 to 9 in. long and up to 4 in. deep, as shown in Figs. 9 and 10. Slant torch cuts are burnt through at *AC*, *AD*, *BC*, and *BD* (Fig. 9), a portion from the corner of the ingot thus being removed. This is cut through *CD* to provide a horizontal section through the corner of the ingot, which section can be polished and either etched or sulphur printed to reveal the sub-surface structure of the ingot.

In order to ensure that this method of sampling will not cause a defect in the subsequently rolled product, angles *ACB* and *ADB* should be large, say about 120°, so that with *AB* equal to 6 in., *AC*, *AD*, *BD*, and *BC* would equal about $3\frac{1}{2}$ in. Examples

Fig. 17—Photomicrograph of seams, showing blow-hole remains. × 100 (Reduced to one-half linear in reproduction)

of sections obtained in this way are shown in Figs. 11 to 14.

The ingot can still be rolled after this treatment without loss, whilst the corner section can then be examined at leisure in the laboratory. Records of defective steel, if any, produced after rolling the ingot can be related to the depth of skin in the original ingot.

The technique is simple and inexpensive and is being further investigated by other members of the Ingot Defects Sub-Committee. Sufficient work has been done to show that the corner sample is reasonably representative of the whole of the periphery of the ingot at the level at which the sample has been taken. Normally, for comparative purposes, the ingot corner is cut at a position half-way up the ingot, but where

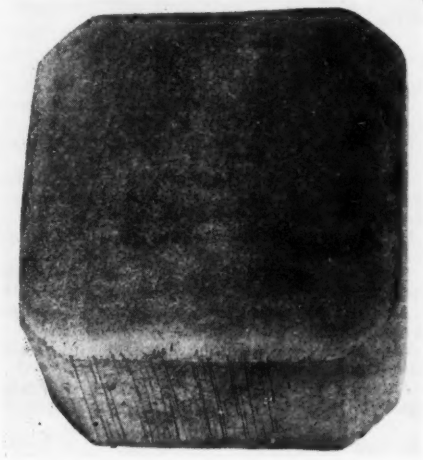

Fig. 16—Seamy billet

Fig. 18—Seamy plate surface. One-half actual size

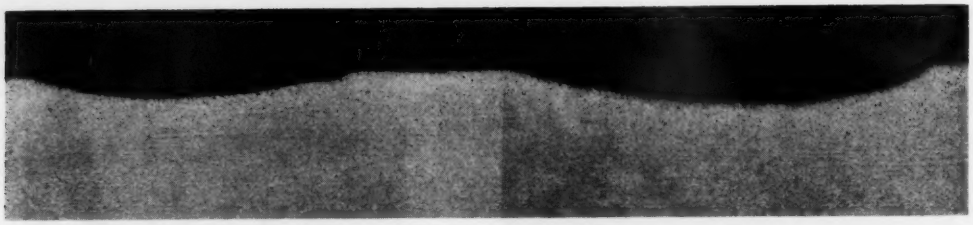

Fig. 19—Seamy plate, cross-section. × 15

further information is required regarding the variation of blow-hole depth along the length of the ingot, samples may be taken at three standard positions 15%, 50%, and 80%, respectively, from the bottom. Typical examples of ingot corner sections at these positions are shown in Fig. 13. This ingot, and more especially the one corresponding to Fig. 11, is relatively thick-skinned. In this case freedom from defects due to sub-surface blow-holes is ensured, as even after soaking and reheating a sound skin is present. The blow-holes are not oxidized, and they roll out satisfactorily, leaving a good surface with a faint trace of rim. A more truly balanced ingot with a shallow line of blow-holes slightly below the surface is shown in Fig. 12. After cogging, the very thin skin and blow-hole remains will be largely washed off in the reheaters before the finished product is produced.

Which type of ingot it is best to aim for in structural steels is a matter for discussion, though on the whole, opinion appears to favour the thicker skin. The worst possible type is where the skin is not thick enough to protect underlying blow-holes and these are too large to wash away completely (Figs. 14 and 15). The result is the production of seamy defects of the type shown in Figs. 16 and 17, which are respectively a macrograph and a micrograph of a seamy billet. The corresponding photographs of a seamy plate are shown in Figs. 18 and 19.

The results obtained in practice will vary from works to works depending upon the precise degree of scaling taking place in soakers and reheaters. At the works with which the writer is associated certain reheaters may give defective steel owing to this trouble, which is practically absent on other reheaters working under different time and temperature conditions.

Overtiming in the soakers may also tend to produce troubles of this type, though in other works and with somewhat different skin depths overtiming has been reported to be advantageous. Care is therefore required in interpreting results, but this consideration does not alter the desirability of a careful study of the factors controlling skin depth in ingots of this type. A statistical study of about 50 ingot corners taken from 10-ton ingots showed that significant correlations could be found between skin thickness and the following factors, skin thickness being greater with :

(*a*) Larger ladle skulls (*i.e.*, colder steel at teeming)
(*b*) Higher aluminium additions to the mould
(*c*) Longer strip to tap time (*i.e.*, cooler moulds)
(*d*) Lower phosphorus content.

The correlation of skin thickness with the first two factors was high, but was only just significant for the latter two. The variation with higher mould aluminium is probably a reflection of a higher degree of oxidation, and it is not suggested that the actual additions to the mould were the cause of the variation ; they were simply a measure of the state of oxidation of the steel. It is hoped to extend the investigation of this point and to include measurements of the oxygen content of the steel.

These factors are probably similar to those controlling skin depth in fully rimming steels, but the range of depth in these structural steels is considerably narrower. Various members of the Ingot Defects Sub-Committee are carrying out similar work, which will no doubt lead to a greater understanding of the factors controlling this important ingot property.

Acknowledgments

Acknowledgments are due to the staff of the Works Metallurgical Department of the Appleby-Frodingham Steel Co., and in particular to Mr. A. Goodall, in charge of the section concerned, for assistance in the investigations described in this paper. Thanks are also due to Mr. Denis Ward, B.A., for assistance in the statistical analysis, to the Melting-Shop and Mill Managements for their encouraging criticism, and to the General Management for permission to publish the results.

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Studies on Anti-Fouling Compositions

By H. Barnes, B.A., B.Sc., Ph.D., F.R.I.C.

FOREWORD

This paper forms part of a series of studies on anti-fouling compositions. The first three papers were published in the December, 1947 issue of the *Journal*. Part I was a general introduction to the whole series. Part II described the probable reactions occurring between the components during preparation of varnish, and Part III gave details of a study of the rate of loss of materials from a series of varnish media immersed in the sea over two consecutive 40-day periods. Part IV, published in the June, 1948 issue of the *Journal*, determined critical leaching rates for a number of common fouling organisms, and compared copper loss under service and raft exposure conditions. The present paper, Part V, gives consideration to the behaviour of mercuric oxide in some anti-fouling compositions, particular attention being paid to the leaching-rate test.

Part V—The Behaviour of Mercuric Oxide in some Anti-Fouling Compositions with Special Reference to the Leaching-Rate Test

SYNOPSIS

The differences in properties, particularly as regards leaching rate of cuprous oxide and mercuric oxide compositions are given some consideration. After examination of a number of possible reasons for this behaviour it is suggested that one of the most important factors concerned is the formation of mercury soaps. The suggestion is made that due to these differences the standard leaching-rate technique cannot be applied to mercuric oxide compositions. The above suggestion has been tested by investigations of the initial leaching rate of mercuric oxide compositions, by analysis of paints, and by studying the behaviour in the presence of zinc oxide as a secondary pigment. The results may readily be interpreted on the hypothesis given.

The effect of mercuric oxide on the properties of the medium is briefly considered. Soap formation appears to give a more resistant varnish medium and one important consequence of this is that the leaching rate of cuprous oxide compositions containing large quantities of rosin can be reduced by the addition of moderate quantities of mercuric oxide.

An examination of the performance of mercuric oxide compositions, together with the data already given, leads to the suggestion that, when leaching as an organic soap, mercury is less effective as a poison than when leaching in the inorganic state.

It is emphasized that this work is of a preliminary nature and serves to indicate the extreme complexity of the systems under investigation.

Introduction

THE greater part of the work being described in this series of papers has been concerned with the behaviour of cuprous oxide in anti-fouling compositions. A similar leaching technique has been used, however, to investigate some compositions containing mercuric oxide, both alone and in conjunction with cuprous oxide. It was soon evident that the behaviour of the two metal oxides, as indicated by the leaching-rate determinations, was different. A study was therefore made of the behaviour of the mercuric oxide compositions in the leaching-rate test, since it was suspected that this would lead to an explanation of the differences observed. It seems logical to report this work as Part V following directly on the discussion of copper leaching rates which has already been given in Part IV.

The technique used for the determination of mercury leaching rates was that already described for copper leaching rates, the mercury in the leachate being estimated by means of dithizone.*

The differences in behaviour between the two metal oxides in similar media may be summarized as follows :

(1) Except perhaps in the later stages of soaking, considerable changes in the mercuric oxide content with a constant varnish component have little effect on the mercury leaching rate obtained. Changes in cuprous oxide content of a similar order give rise to large changes in the copper leaching rate (*see* Table 1).

The compositions 214–217, which had varnish media based on rosin–bedesol combinations, had no other poison, but the same phenomenon is observed in compositions containing both cuprous and mercuric oxides. Table II gives the mean leaching rates of a number of such compositions over the first sixty days' soaking.

(2) There is frequently a distinct rise in the leaching rate of mercuric oxide compositions in the later stages of soaking at the time when the

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Dr. Barnes, formerly Investigator to the Marine Corrosion Sub-Committee of The Iron and Steel Institute, is now with the Scottish Marine Biological Association.

* H. BARNES : *Journal of the Marine Biological Association of the United Kingdom*, 1946, vol. 26, pp. 303–311.

Table I

LEACHING RATES OF MERCURIC OXIDE COMPOSITIONS

Constant total pigment content = 66·7%
Rosin/binder ratio 4/1

Composition No.	Mercuric Oxide, %	Leaching Rates at Days' Soaking, μg/sq.cm./day					
		3	10	30	46	57	83
217	2·6	2·2	1·6	1·3	1·5	1·3	0·9
216	5·1	2·5	1·1	1·4	1·3	0·9	1·1
215	7·7	4·4	1·0	1·7	2·0	1·8	...
214	10·2	2·2	1·6	1·8	1·9	2·1	...

leaching rate of a similar cuprous oxide composition would be tending to decrease.

(3) A marked contrast is found in the actual leaching rates of cuprous oxide and mercuric oxide compositions of comparable formulation and poison content. For example, the mean leaching rate over 15–60 days of a 7·7% cuprous oxide composition was 6·3 μg Cu/sq.cm./day whilst for 7·7% of mercuric oxide paint in the same medium it was 1·6 μg Hg/sq.cm./day. It is possible that this is due to differences in the rate of reaction or solution, of the two oxides with sea water, but experiments using cuprous oxide and mercuric oxide 'surfaces' (*see* p. 180) have indicated no differences of this order in the rate of reaction of the two oxides.

(4) From time to time, paint from leaching slides or other panels has been analysed after soaking and the actual mercury losses determined. Some results of these determinations for compositions containing 15·4% of Cu_2O are as follows :

Composition No.	HgO, %	Leaching Rate Graph (mean)	Rate of Loss, Analysis	Ratio of Rates
18H	2·6	1·0	1·3	0·77
19H	5·1	1·1	1·8	0·61
20H	7·7	1·4	2·3	0·65

These results indicate two important features of these compositions. Firstly, according to the analytical figures, equal additions of mercuric oxide produce similar rises in the actual rate of loss of mercury (an addition of 2·6% gives a rise of approximately 0·5 μg/sq.cm./day) and this is in contrast to the small effect on the leaching rate. Secondly, unlike cuprous oxide compositions (*see* Part IV), the loss as estimated by analysis is greater than that indicated by the leaching-rate determinations.

(5) The physical properties of compositions containing mercuric oxide are somewhat different from those of cuprous oxide compositions ; the former are more viscous, have different flow properties and, when dry, are more brittle.

Table II

MERCURY LEACHING RATES FOR CUPROUS OXIDE/MERCURIC OXIDE COMPOSITIONS

Constant total pigment content = 66·7%
Rosin/binder ratio 4/1

Composition No.	Cu_2O, %	HgO, %	Mean Leaching Rate, 15–60 days, μg/sq.cm./day
14H	7·7	2·6	0·7
15H	7·7	5·1	0·9
16H	7·7	7·7	1·5
18H	15·4	2·6	1·1
19H	15·4	5·1	1·0
20H	15·4	7·7	1·4
22H	23·0	2·6	1·1
23H	23·0	5·1	1·1
24H	23·0	7·7	1·1
26H	30·7	2·6	1·5
27H	30·7	5·1	2·2
28H	30·7	7·7	1·6

THE LEACHING RATE OF MERCURIC OXIDE COMPOSITIONS

It is convenient to consider at first the fourth of these differences. Experience with cuprous oxide compositions suggests three possibilities which might account for the lower value of the leaching rate when compared with actual loss determined by analysis.

It is possible that 'losses' of mercury occur in the leaching test owing to adsorption of the mercury on the suspended or colloidal material in the sea water, or to volatilization or other causes, for such 'losses' of mercury have been found to occur in sea water solutions of mercuric salts when the latter are allowed to stand for some time.* It has been shown, however, that no significant losses occur in test solutions under the conditions of the leaching-rate test.

An alternative possibility is that the whole of the mercury passing from the paint to the leachate is not estimated by the analytical technique used. This technique is based on a mercuric chloride calibration curve, and the solution of small amounts of mercuric oxide in sea water would be expected to give the same results as mercuric chloride itself in sea water. It is possible, however, that, if the mercury were reduced in the paint film and were leached in the mercurous form, or if combination to form mercury soap took place and this substance were being dissolved, a correct return of mercury would not be obtained. With the small amounts present it is a matter of some difficulty to establish by direct methods the validity or otherwise of such assumptions. Indeed, it is difficult to prepare mercury rosinate in a form sufficiently pure for the purpose of such an investigation. Experiments with mercury stearate suggest that no 'loss' of mercury should take place if the mercury is in the form of a combination with a fatty

* N. W. Hanson and R. F. Jervis : *Journal of the Society of Chemical Industry*, 1946, vol. 65, Nov., pp. 329–333.

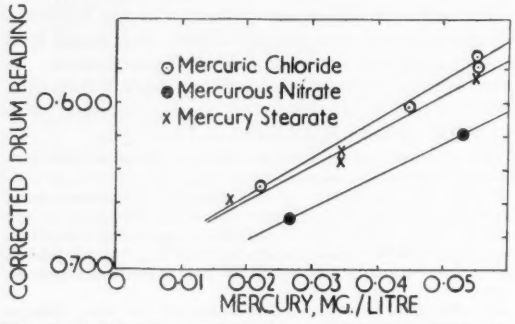


Fig. 1—Calibration curves for various mercury compounds

Table III

ACCUMULATION OF COPPER IN LEACHING BATH

Cuprous oxide composition

Time, hr	Copper Concentration in Leachate, mg/litre
2	0·20
4	0·42
8	0·80
12	0·99
16	1·29
20	1·36
24	1·43
48	1·50

acid. Figure 1 shows mercury calibration curves obtained with mercuric chloride and mercury stearate (a commercial product whose mercury content was determined by analysis). It would seem that when present as stearate the mercury would be accounted for by the analytical procedure used; the difference in the region of concentration found in leachates, *i.e.*, up to 60 μg/litre, is only in error by about ± 5 μg/litre. A calibration curve for a mercurous salt (mercurous nitrate) is also shown in Fig. 1. It is clear that should the mercury be leaching in the form of a mercurous salt considerable error would be involved if the leaching rate were interpreted from a mercuric chloride calibration curve; amounts up to 60 μg/litre would have a negative error of approximately 30% throughout the whole range. General considerations render unlikely this latter possibility, and while the subsequent work indicates that it is not, in the first instance, of primary importance, its possible significance in the interpretation of the behaviour of mercury-containing paints must be remembered, particularly when these paints are stored.

It has been pointed out (*see* Part I) that an essential condition in the leaching-rate test is that no retardation of the rate of loss should occur by accumulation of the products in the leachate. Up to a concentration of 0·5 mg/litre (Table III and Fig. 2) the rate of loss is certainly constant for cuprous oxide compositions.*

If a copper compound with a much smaller solubility in sea water than cuprous oxide is used then this condition is not fulfilled in the 4-hour test. The data for a copper carbonate composition (*see* Table IV) show that the rate of loss with such a composition falls off rapidly; the true leaching rate can only be obtained by determining the slope of the tangent to the time-concentration curve at the origin (*see* Fig. 3).

Clearly in the latter case the 4-hour leaching rate as usually determined is not comparable with that obtained for a cuprous oxide composition and similar

* The leaching rate obtained when these conditions are fulfilled will be termed "the true leaching rate of the composition."

Table IV

ACCUMULATION OF COPPER IN LEACHING BATH

Copper carbonate composition

Time, min	Copper Concentration in Leachate, mg/litre
5	0·010
10	0·015
15	0·019
35	0·023
60	0·032
120	0·040

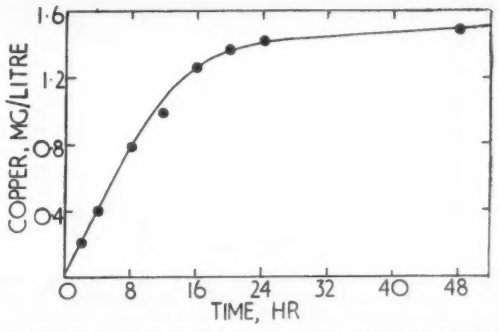


Fig. 2—Accumulation of copper in leaching bath using a cuprous oxide composition

results would always be obtained with paints containing poisons whose solubility in sea water was very small. If, therefore, in the leaching of a paint containing mercuric oxide a similar phenomenon to that with copper carbonate paint occurs, the true leaching rate would not be obtained by the usual technique.

In order to test this possibility a number of slides coated with a mercuric oxide composition 191H were leached (after being soaked for some weeks) for various periods of time, with the results shown in Table V and Fig. 3.

Table V

ACCUMULATION OF MERCURY IN LEACHING BATH

(191H)

Total pigment = 66·67%
Mercuric oxide = 15·38%
Rosin/Binder ratio = 4/1

Time, min	Mercury Concentration in Leachate, mg/litre
12	0·011
28	0·016
42	0·020
120	0·041
202	0·073
306	0·080

The graph is not linear over the first four hours so that the 4-hour leaching rate cannot have the meaning usually ascribed to it.

THE SUBSTANCE LEACHED FROM MERCURIC OXIDE COMPOSITIONS

Since mercuric oxide is comparatively soluble in sea water even when compared with cuprous oxide, it seems unlikely that such a slowing down of the rate of solution would be obtained if mercuric oxide

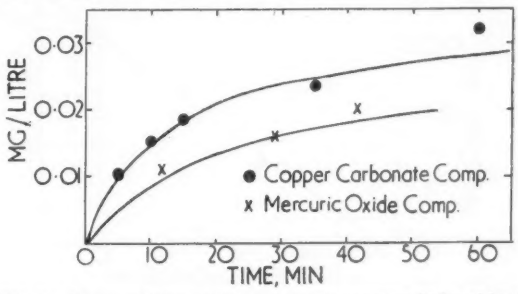


Fig. 3—Accumulation of copper and mercury in leaching bath

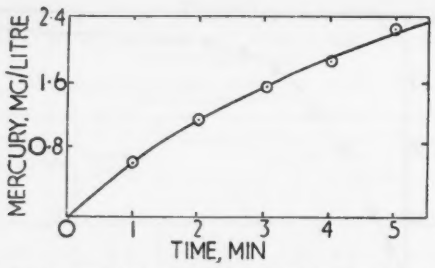


Fig. 4—Accumulation of mercury in leaching bath using 84% mercuric oxide composition

itself were dissolving. To test this point a paint was made containing 84% of mercuric oxide in a resistant varnish. In such a composition there is continuous contact between the surface particles so that leaching takes place from a mercuric oxide surface. The concentration of mercury in the leachate, when such a paint was leached without any previous soaking, is shown below and in Fig. 4 :

Time, min	Mercury Concentration in Leachate mg/litre
1	0·69
2	1·16
3	1·59
4	1·86
5	2·26

It is quite clear that the solubility of mercuric oxide in sea water is unlikely to be a limiting factor in the leaching of mercuric oxide compositions when it is remembered that linearity was lost at values below 0·06 mg/litre with a normal mercuric oxide paint.

On the basis of this evidence it seems that the substance being leached is not entirely mercuric oxide. It is suggested that in the milling of these compositions much of the mercuric oxide combines with the medium to give mercury soaps, and that the low solubility of these substances in sea water causes the rapid slowing down of this rate of solution.

EVIDENCE FROM THE INITIAL LEACHING RATE

The lack of any correlation between mercuric oxide content and the leaching rate (p. 178) is, on this basis, due to the fact that the true leaching rate is not determined by the standard leaching technique.

If the leaching rates are determined over short periods, the initial rates should then be proportional to the mercuric oxide content of the compositions.

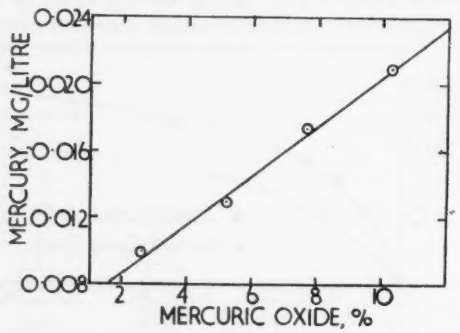


Fig. 5—Concentration of mercury in leaching bath with varying mercuric oxide content (20 min value)

The amounts of mercury leached in the early periods, below, say, 20 min, are so small that accurate analysis is difficult. If the values of the 20 min leaching are taken, then the leaching-rate/mercuric-oxide content graph is linear above 2·56% of mercuric oxide content as shown below* and in Fig. 5 :

Mercuric Oxide, %	Mg/litre in 20 min
2·56	0·010
5·12	0·013
7·68	0·017
10·24	0·021

EVIDENCE FROM ANALYSIS OF WET PAINT

That a reaction takes place when mercuric oxide is milled with the media used is indicated by the thickening of the composition which usually takes place. An attempt has been made to separate the components of such a composition and to determine the amount of free mercuric oxide. The rosinate is not very soluble in most organic solvents and there is some difficulty in effecting a separation.

The paint sample (191H) was repeatedly washed and centrifuged with an alcohol–acetone–chloroform mixture, and the residue weighed. The residue was then analysed, with the following results :

Wet Paint	Original HgO	Residue	HgO in Residue
283·1 mg	34·0 mg	127·2 mg	15·0 mg

From 283·1 mg of wet paint containing 34 mg of HgO and 113 mg of ferric oxide the weight of pigments left was 127 mg. Had all the mercuric oxide and iron oxide remained in the separated pigments it would have weighed 147 mg. If the loss, 20 mg, is assumed to be mercuric oxide reacted with rosin to give the rosinate, and removed in the solvent, this agrees with the analysis of the residue which gave a mercuric oxide content of 15 mg or a loss of 19 mg. (This separation is not as satisfactory as these figures suggest, and the excellent agreement is to some extent fortuitous). They indicate, however, that some 60% of the mercuric oxide has reacted with the rosin and, on the assumption of the formula $Hg(Ab)_2$ for the rosinate, this means that more than 95% of the rosin has reacted.

It has been assumed in the foregoing that mercuric oxide either reacts to form the soap or remains unchanged. It is realized that further reactions may take place, and the reduction of some oxide to metal seems probable. The residue has been assumed to be HgO in the above calculations, but, by the analytical method used, any metallic mercury could be included as mercuric oxide.

The formation of soaps with the mercuric oxide need not be considered as being limited to the reaction with rosin ; other components of the medium may react. Indeed the fact that a more stable matrix is apparently produced with the media under investigation, in conjunction with the fact that mercury rosinate is not usually considered a particularly stable

* It should be emphasized that the amounts of mercury in these and other leachates discussed are very small—since 60 ml of leachate are available from one test. The limit of reasonably accurate work is being approached and, for more detailed investigations, an analytical technique capable of dealing with smaller concentrations of mercury is required.

substance, suggests interaction with other varnish components. An alternative possibility is that mercury rosinate is stabilized to a greater extent by the binder used than the rosin itself is stabilized.

It should also be noted that if, as these results suggest, the rosin has been converted into the mercury soap, the mercury is being leached not only from unreacted mercury (oxidized or reduced to metal) but also from the medium, *i.e.*, mercury soap. The insolubility of the latter, in addition to giving a spurious mercury leaching rate *per se*, will affect the leaching rate obtained from any mercuric oxide or metallic mercury remaining as particles.

EVIDENCE FROM ZINC OXIDE/MERCURIC OXIDE COMPOSITIONS

The addition of zinc oxide to mercuric oxide compositions is frequently made by manufacturers and earlier experiments had suggested that its action was to raise the mercury leaching rate. If the reduction in leaching rate was due to the formation of mercury soap it is reasonable to suppose that the introduction of an oxide with a considerable affinity for rosin prior to the introduction of mercuric oxide would reduce interaction of the latter substances, and that the consequent reduction in the mercury soap would raise the mercury leaching rate of the resultant composition. To test this hypothesis a series of paints was prepared containing zinc oxide and mercuric oxide on the basis of a total pigment content of 66·7% and a 4/1 rosin/binder ratio. Series 1 (213–216) had graded mercuric oxide content replacing an equivalent quantity of iron oxide ; Series 2 (218–221) had the same quantities of mercuric oxide and a constant amount (12·8%) of zinc oxide milled together ; Series 3 (223–226) had the same formulation as Series 2 except that the zinc oxide was first milled 12 hours with the medium, after which the mercuric oxide was added and the milling continued for a further 12 hours. The leaching rates of these compositions are shown in Table VI.

The pre-milling with zinc oxide and the consequent interaction might be expected to produce zinc rosinate, leaving more free mercuric oxide after the second milling on addition of this poison. This has two possible effects : firstly, it gives a new component (zinc rosinate), which is possibly less resistant to sea water than mercury rosinate and which would, by its greater rate of breakdown, tend to give a higher leaching rate for any contained particulate poisons. That a rise in leaching rate is produced by this process is indicated by the values for the mercury leaching rate of Series 3 (*see* Table VI). Secondly, it would also leave more mercuric oxide unreacted and, therefore, the mercury leaching rate obtained from such compositions should approximate to a true leaching rate. This latter is more critical for the hypothesis put forward, *i.e.*, the relationship of the mercury concentration in the leaching bath and the time of leaching. If, in Series 3, the zinc oxide has removed most of the rosin in pre-milling, the mercury should now be present as oxide and the 'anomalous' behaviour be eliminated.

Table VI

LEACHING RATES OF MERCURIC OXIDE/ZINC OXIDE COMPOSITIONS

	HgO, %	Composition No.	Leaching Rate at Days' Soaking, μg/sq.cm./day			
			10	30	46	57
Series 1, HgO only, no ZnO	5·1	216	1·1	1·4	1·3	0·9
	7·7	215	1·0	1·7	2·0	1·8
	10·2	214	2·2	1·6	1·8	1·9
	12·8	213	1·5	1·3	1·3	0·9
Series 2, HgO milled + 12·8% ZnO	5·1	221	2·3	2·4	2·4	2·3
	7·7	220	1·7	3·1	3·0	3·2
	10·2	219	2·1	2·8	2·4	2·4
	12·8	218	1·6	1·6	2·4	1·9
Series 3, 12·8% ZnO milled in first	5·1	226	2·9	3·4	3·0	2·4
	7·7	225	4·4	8·6?	5·7	3·7
	10·2	224	7·8	9·6	8·0	5·2
	12·8	223	10·4	11·5	10·0	7·2

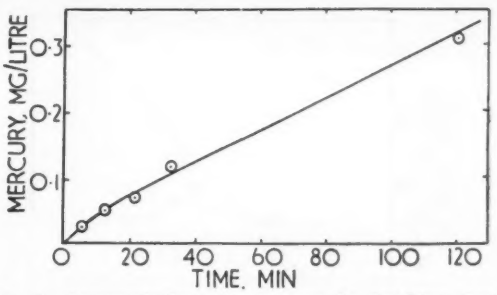


Fig. 6—Accumulation of mercury in leaching bath using mercuric oxide/zinc oxide composition (ZnO milled with constituents before addition of HgO)

The accumulation of mercury in the leaching bath was followed for a paint from Series 3 with the results shown below and in Fig. 6 :

Time, min	Mercury Concentration, mg/litre
5	0·028
12	0·053
21	0·074
33	0·117
120	0·309

Except in the early portion of the curve the relation is linear.

In addition, in Fig. 7, the mean 4-hour leaching rates for Series 3 (30–57 days) are plotted against

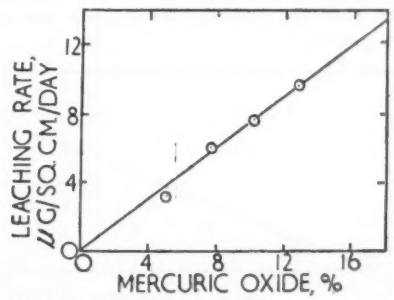


Fig. 7—The mean leaching-rate/mercuric oxide content at 30–57 days of soaking (Series 3—pre-milled with ZnO)

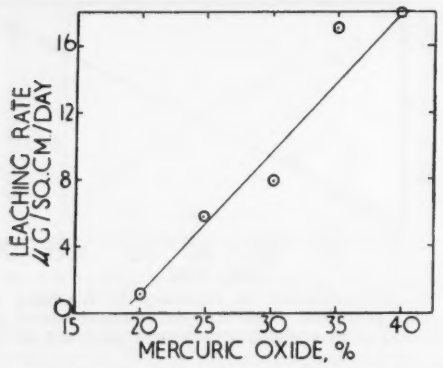


Fig. 8—Variation of leaching rate with mercuric oxide content (high mercuric oxide composition)

the mercuric oxide content; the leaching rate is seen to be proportional to the mercuric oxide content when the zinc oxide was milled in first.

EVIDENCE FROM HIGH MERCURIC OXIDE COMPOSITIONS

The formation of mercury soap is, of course, limited by the amount of acidic material present, although the limit need not be that of the usual stoichiometric proportions. (Other work has shown that more of some oxides can be 'reacted' with more rosin than is thus indicated.) Since 217 g of mercuric oxide are equivalent to 602 g of rosin, a mercuric oxide composition based on the same formula as compositions 213–226 would have all the rosin reacted at about 9% mercuric oxide content. Above this, the leaching rate of the compositions might be expected to rise rapidly with increasing mercuric oxide content and to give leaching rates proportional to the excess mercuric oxide.

Compositions 269–273 were based on the same formula as 213–226 with graded amounts of mercuric oxide from 20–40%. Their leaching rates are shown in Table VII, and the rate at 31–42 days of soaking plotted against mercuric oxide content in Fig. 8.

It appears that the leaching rate was retarded during the initial days of soaking, but Fig. 8 indicates that above 20% of mercuric oxide content the mean leaching rate for 31–42 days is proportional to the HgO content. This suggests that the rosin can react with more than the equivalent amount of mercuric oxide or that the varnish components are reacting with the oxide.

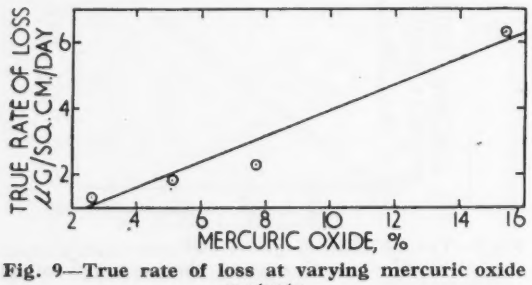


Fig. 9—True rate of loss at varying mercuric oxide contents

LEACHING RATE AND TRUE RATE OF LOSS

The rate of loss in the sea was determined for some of the panels of cuprous oxide and mercuric oxide compositions exposed in 1944. These together with 191H are shown below:

Composition No.	HgO, %	Cu_2O, %	Rate of Loss, Hg by Analysis (122 days), µg/sq.cm./day
18H	2·6	15·4	1·3
19H	5·1	15·4	1·8
20H	7·7	15·4	2·3
191H	15·4	0·0	6·3

No data are available for the true leaching rates of the sea-soaked slides of this series, but the relationship between the true rate of loss and the HgO content is shown in Fig. 9. From 2·6–15·4% of mercuric oxide the rate of loss increases proportionally to the increase of mercuric oxide content which would be expected if all the oxide was in the form of rosinate.

The greater loss of mercury when zinc oxide is milled into a paint was also confirmed by analysis of the paint from panels. The analyses were made from panels which had been exposed on the raft and were made at too late a date to show more clearly the contrast suggested. The compositions were those already described (Series 1, 2, and 3, compositions 213–227) and the results are as follows:

Composition No.	HgO, %	Rate of Loss, µg sq.cm./day	Loss, %
223	12·8	27·0	82
225	7·7	14·2	84
218	12·8	16·7	48
220	7·7	11·5	57

The first of these compositions had 12·8% zinc oxide milled in first, while the second pair had the mercuric oxide milled in first.

THE EFFECT OF MERCURIC OXIDE IN THE MEDIUM

Soap formation has a distinct effect on the medium of a composition. Due to the reaction, the acidity of the medium is suppressed with the production of

Table VII

LEACHING RATE OF MERCURIC OXIDE COMPOSITIONS

Composition No.	HgO, %	Leaching Rate at Days' Soaking, µg/sq.cm./day				
		3	10	21	31	42
269	20	1·3	0·5	0·2	0·7	1·6
270	25	2·6	1·2	1·7	5·5	6·0
271	30	2·9	1·4	3·6	8·1	7·3
272	35	10·3	7·3	12·9	17·9	16·0
273	40	9·8	7·3	13·1	20·5	15·9

apparently more resistant matrix. By analogy, with the effect of increasing the binder proportion in cuprous oxide compositions it would be expected that such a change would tend to reduce the copper leaching rate in cuprous oxide/mercuric oxide compositions. Data are available for the actual loss of varnish components. Slides have been weighed

before and after soaking and the total loss in weight determined; analysis for pigments has been made and the loss of varnish computed by subtraction. The results for two compositions, 17H and 191H are shown in Table VIII. (Film breakdown made the later values for 191H unreliable). The formulations of these two compositions were identical, except that in 191H mercuric oxide replaced the cuprous oxide of 17H.

The addition of mercuric oxide has reduced the rate of loss of matrix. The effect on copper leaching rate is indicated by Fig. 10 which shows the leaching rate curves for 21H, 23H and 24H with 23·0% of cuprous oxide, and 0·0, 5·1, and 7·7% of mercuric oxide, respectively. Two effects of the added mercuric oxide may be noted.

Firstly, the high early leaching rate of the cuprous oxide composition has been much reduced, and the leaching curve shows an initial rise. This would appear to be due to the effect of mercuric oxide on the accumulation of copper at the surface and indicates that these cuprous oxide/mercuric oxide compositions are of such a nature that little surface cuprous oxide is exposed. This may be due to the coating of the cuprous oxide particles by the mercury soap so that ready solution does not take place, or it may be due to the effect of the changed nature of the medium on the processes taking place when the paint dries out, preventing the production of exposed cuprous oxide particles. This phenomenon is of considerable practical importance. If, for example, the steady value of 24 μg/sq.cm./day leaching rate of these paints (C–D of curve) were required for anti-fouling then the paints would not be anti-fouling for the first 30 days after immersion and, if subjected to intense fouling conditions, failure would ensue. This phenomenon seems to have been evident in the paints 23 and 28H used for recent service tests. Thus control panels of 28H exposed in October 1944 fouled almost immediately with *Tubularia* (of which there was a reasonably heavy settlement at this time); from this fouling the plates never recovered. An exposure of the same composition in January 1945, when few

Table VIII

LOSS IN WEIGHT PER SLIDE (12·9 SQ.CM.) AFTER SOAKING

Days	17H		191H	
	Total Loss, wt, mg	Total Cu Lost, mg	Total Loss, wt, mg	Total Hg Lost, mg
30	21·0	9·8	5·4	2·6
60	...	14·7	9·7	5·2
90	53·2	17·5	13·2	7·6
120	62·7	19·2	...	...
150	70·5	20·0	...	...

important fouling organisms were settling, remained clean during this season.

MERCURIC OXIDE COMPOSITIONS AND ANTI-FOULING

The actual rate of loss in 191H with 15·4% of mercuric oxide has been shown to be 6·5 and 6·0 μg/sq.cm./day during the first and second 60 days, respectively. The main fouling data for two exposures of this composition together with that of the non-toxic plates exposed during the same period are shown in Table IX.*

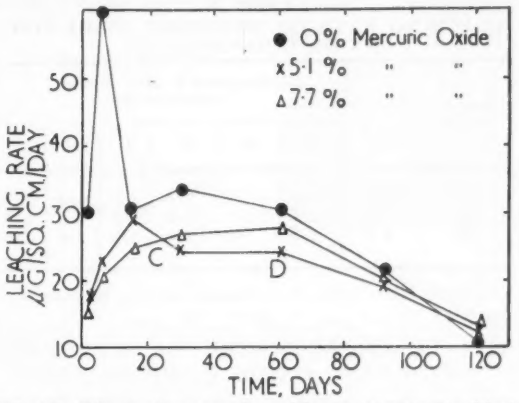


Fig. 10—Effect of mercuric oxide on copper leaching rate (18% cuprous oxide)

In the first exposure the paint failed to the hydroid *Tubularia* after 65 days and by 95 days had accumulated some mussels. In the second exposure it failed to barnacles at 90 days. It seems clear, therefore, that a rate of loss even of 6·0 μg/sq.cm./day is inadequate to prevent settlement of these two fouling forms. Other evidence indicates that when leaching in the form of inorganic mercury takes place the critical leaching rate is of the order of 2 μg/sq.cm./day. If this is so then the maximum efficiency of mercury as an anti-fouling agent is not being obtained in these compositions in which much of the mercury has reacted, and in which it is leaching in the form of soap. On this evidence the mercury seems to be

* Data supplied by Mr. K. A. Pyefinch.

Table IX

RAFT PERFORMANCE OF SPECIMEN 191H

15·4% of mercuric oxide as only poison

First Exposure:	31 days		65 days		95 days	
191H (*a*)	Sl.	0·25	*Tu.*	4·0	*Tu.*	4·0
	Ob.	trace			*Myt.*	trace
	Tu.	trace				
191H (*b*)	Sl.	0·25	Sl.	0·25	*Tu.*	2·0
			Tu.	2·0		
Non-toxic (*a*)	*Tu.*	1·0	*Tu.*	4·0	*Tu.*	2·0
	Ob.	trace			*Myt.*	1·0
Non-toxic (*b*)	D.	4·0	*Tu.*	2·0	...	
	P.	2·0	*Ect.*	3·0		
			Ent.	1·0		
Second Exposure:	29 days		65 days		90 days	
Plate 1, 191H	Clean		Sl.	0·5	D.	2·0
					B.	4·0
Non-toxic	Clean		D.	3·0	*Ect.*	3·0
			Tu.	trace	*Ent.*	3·0
					Tu.	trace
					B.	6·0

Sl. = Slime; *P.* = *Pomatoceros*; *Ect.* = *Ectocarpus*; *Ob.* = *Obelia*; *Myt.* = *Mytilus*; *Ent* = *Enteromorpha*; *Tu.* = *Tubularia*; B. = Barnacles; D. = Diatoms

Table X

LEACHING RATE OF MERCURIC OXIDE/ZINC OXIDE COMPOSITIONS

Composition No.	HgO, %	Leaching Rate at Days' Soaking, μg/sq.cm./day					
		3	10	21	31	42	62
223	12·8	14·9	14·8	7·5	3·8	4·4	1·9
225	7·7	6·4	14·9	10·6	1·3	1·6	1·5
227	2·6	1·3	1·6	1·6	0·9	1·0	1·0

less toxic than when it is leached out as an inorganic complex.

THE LEACHING RATES OF MERCURIC OXIDE/ZINC OXIDE COMPOSITIONS

It has been indicated that in Series 3 of these compositions, in which the zinc oxide was pre-milled with the varnish, much of the mercuric oxide remains unreacted and probably a true mercury leaching rate is obtained. The leaching rates should, therefore, give a true indication of the processes taking place in the sea and should be directly related to the anti-fouling performance.

Data are available for three compositions of this series—223, 225, and 227 with 12·8, 7·7, and 2·6% of mercuric oxide, respectively. The leaching curves up to 62 days are shown in Fig. 11 and the data are also given in Table X. It should be noted that two of these curves are of the same form as those for copper leaching curves for a cuprous oxide composition using the same medium (contrast p. 183).

All the panels became fouled with *Tubularia* at 63 days, but composition No. 223 had only a trace of this hydroid. This suggests that a leaching rate of 1·9 μg/sq.cm./day is inadequate to prevent settlement of this resistant form, but that it is not far removed from the critical value. This value is to be contrasted with the ineffective value of 6·0 μg/sq.cm./day (true rate of loss) against *Tubularia* for 191H (*see* p. 183), a value which would correspond to a 'four-hour leaching rate' of about 7·0 μg/sq.cm./day.

Further, the non-toxic control panels at 63 days, in addition to a significant rating of *Tubularia*, had *Ectocarpus*—trace, *Obelia* 1·0, and *Pomatoceros* 1·0, indicating the availability of these organisms. It appears that these latter organisms, since at this stage they were absent from the panels painted with all three compositions, are prevented from settling by a true mercury leaching rate of the order of 1·0 μg/sq.cm./day. It is essential to have much more information before the real critical leaching rate of mercury can be accurately assessed.

The present investigations, while to some extent clarifying the problems associated with compositions containing mercuric oxide, have emphasized the complexity of the system under consideration. Little has been said concerning the possibility of free mercury in these compositions, but some is almost certainly present, initially, and may increase on storing the composition. Systems containing rosin and mercuric oxide are also distinctly sensitive to light—apparently with reduction of the oxide. Whilst mercury rosinate may predominate in the

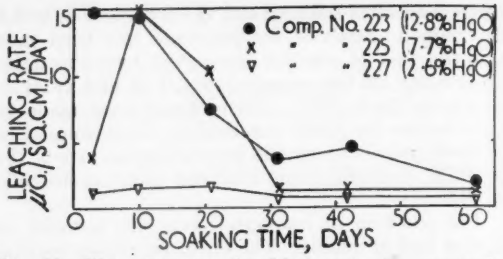


Fig. 11—The mercury leaching rate in mercuric oxide/zinc oxide compositions

compositions studied there may then be, in addition, mercuric oxide, metallic mercury, and mercurous compounds and changes in conditions or formulation may bring about alterations in this respect. The production of these substances from mercuric oxide affects not only the poisonous pigment concentration but also its nature; at the same time changes in the matrix of the composition take place due to soap formation and oxidation. To elucidate the contribution of these to the observed leaching rates, which in any event are quite small, will require considerable ingenuity; to relate them to the sequence of events on an exposure panel will be no less difficult.

SUMMARY

A consideration of the behaviour of cuprous oxide and mercuric oxide compositions indicates that in a number of important respects they stand in marked contrast. Thus:

(1) Considerable changes in the mercuric oxide content with a given medium give little change in leaching rate; similar changes in cuprous oxide content with the same medium would give rise to very different rates.

(2) The mercury leaching rate tends to rise, copper leaching rate to fall, in the later stages of exposure.

(3) For the same poison content in the same medium the copper and mercury leaching rates are widely different.

(4) The loss of mercury indicated by leaching rate determinations is less than the actual loss as determined by analysis; in cuprous oxide compositions the reverse is the case.

(5) The two types of composition differ in physical properties.

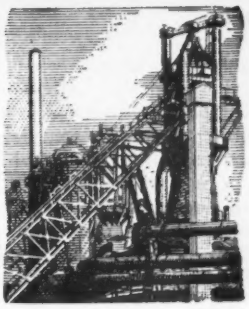
The possible reasons for these differences in behaviour are examined and it is suggested that the formation of mercury soap plays an important part. Evidence in support of this is adduced from the initial leaching rates of mercuric oxide compositions, from paint analyses and from the behaviour in the presence of zinc oxide as a secondary pigment.

Some consideration is given to the effect of mercurous oxide on the medium; there is a tendency to produce a more resistant medium with these bedesol varnishes, and the incorporation of mercuric oxide into cuprous oxide compositions containing large quantities of rosin reduces considerably the copper leaching rate both initially and in the later stages.

Some of the results quoted suggest that when leaching in the form of a soap complex mercury is less efficient as a toxic agent than when it is in the inorganic state.

IRON AND STEELWORKS ENGINEERING

THE IRON AND STEEL ENGINEERS GROUP



Belt Conveyors, Bunkers, and Chutes for Iron and Steel Works

By J. Brimelow, B.Sc.(Eng.), M.I.C.E.

SYNOPSIS

A description is given of the chief items used in belt-conveyor construction. The manufacture of the belt is briefly described and reference made to the method of specifying the component parts. The effect of the shape of the troughing idler on capacity and guiding is discussed, and the trend of modern design is noted.

In the design of conveying plant the necessary width of belt, as affected by the maximum lump size or by the thickness of the belt required for power transmission, is considered. The carrying capacity of belts in relation to their width is given, also the maximum speeds used, together with a combined expression for maximum conveying capacity. The estimation of horse-power, belt tension, and the effect of belt stretch at bend and drive pulleys are discussed.

Bunkers are briefly mentioned, and also the use of bottom and side discharge gates for materials of different qualities.

Introduction

BELT conveyors have been used for many decades. At the outset they were used chiefly in comparatively narrow widths to handle such materials as grain and coal, but at the present time there is hardly any material which cannot be transported with ease and economy by means of belt conveyors, which are now regularly made in widths up to 60 in or more.

While improvements in the design of conveyor equipment and belts have been going on, so the work of the designer of the complete conveying apparatus has been modified accordingly, in order that full advantage of the better components at his command may be taken. However, the published literature on the subject has not kept pace, and it is hoped that the following remarks will prove of interest and assistance to those users of conveyor equipment who wish for reliable information for the preparation and consideration of any scheme involving the use of this type of plant.

THE CONSTRUCTION AND OPERATION OF BELT CONVEYORS

The conveyor belt is carried on a series of groups of pulleys generally referred to as 'idlers.' These are of two kinds: one the troughing idler used for the upper or carrying strand of the belt, the other the flat idler used on the return or empty run of the belt. In addition, pulleys, generally referred to as the head pulley, tail pulley, bend pulley, snub pulley, etc., are required at the two ends of the conveyor, and frequently at other points between. Usually the head pulley is connected through suitable gearing to the driving motor, though frequently the drive pulley is placed at some other point.

The whole of this equipment is mounted on a structure generally of steel, but occasionally of timber or concrete, and chutes are provided—one to feed the material on to the belt at the tail end, and one to remove it at the head end, or if required, at an intermediate point. It is the duty of the designer to arrange all these component parts in such a manner that the resultant conveying apparatus successfully carries out the duty required of it.

In order to appreciate the various points which must be considered in the design of a conveyor, it is desirable to have some idea of the construction and design of the parts used, and the more important of these will be briefly considered.

The Rubber Belt

The belt, the most important and most expensive part of the equipment, may be made of various materials. For light duties, belts of cotton, canvas, or balata are satisfactory, but for heavier work, the

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only type of belt which need be considered is the rubber conveyor belt, which, when used in conjunction with a well-designed layout, may be relied upon to give good service as measured by the total tonnage carried throughout its life.

Construction—The body of the belt, which takes the main tension of the conveying force, is formed of cotton duck, woven from strands of cotton fibres. The duck is made in widths of about 6 ft and in lengths up to about 600 ft. The lengthwise threads (warp), which carry the belt tension, are more closely spaced than the crosswise threads (weft).

Cotton duck is specified by giving the weight in ounces of a piece of the material 42 in long (warp direction) by 36 in wide. Most conveyor belts are made in standardized weights of 28-oz or 32-oz duck, though for heavier duties, ducks of 36 oz and 42 oz may be supplied. The heavier the duck, the greater the load it can carry.

Before the duck is used in the manufacture of conveyor belts, it is carefully dried by passing it through heated rolls, and then thoroughly impregnated with a special rubber compound or 'frictioning,' which is pressed through the canvas so that the

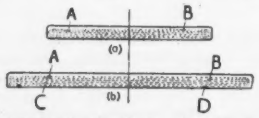

Fig. 1—Arrangement of plies. (*a*) In a narrow belt ; (*b*) In a wide belt (joints shown at A, B, C, D)

separate threads are thoroughly coated. The body of the belt is then built up of layers or plies of the duck, the exact arrangement depending on the width of belt and the number of plies it is to contain. Special care is necessary to arrange that the longitudinal joints in the plies are not too near the edges of the belt.

A typical arrangement of the layers or plies in a comparatively narrow belt is shown in Fig. 1 (*a*).

In this case two separate plies are arranged at the centre of the belt. These are covered by two folds of duck arranged, in the one case, with the edges butting at *A*, and in the other at *B*, both of which points are at some distance from the edge of the belt.

Figure 1 (*b*) shows how the plies might be arranged in the manufacture of a wider belt, in which the width of duck as ordinarily manufactured is not sufficient to form a complete cover to the inner plies as shown in Fig. 1 (*a*).

When the carcase of the belt has been made, it is covered with a layer of rubber, the thickness of which, on the upper or carrying surface, is made to suit the duty of the belt. That on the underside is less thick as it does not have to withstand the same amount of wear. The whole belt is then subjected to a vulcanizing process.

The attention of the belt manufacturers during the past few years has been directed towards improving the rubber cover which takes the wear, and the frictioning compounds used to impregnate the cotton duck. Various qualities of these are now available, and the designer of conveying equipment is able to specify the most suitable belt for any particular duty.

The British Standards Institution issued a British Standard for Rubber Conveyor and Elevator Belting No. 490 dated 1933, which gave not only recommendations as to the positioning of longitudinal and transverse splices in the separate plies, but also further useful information, such as the maximum and minimum numbers of plies desirable in belts of definite widths, the minimum diameters of pulleys recommended for belts of different thicknesses, the desirable width of pulley face, and clearances between edge of belt and any part of the supporting structure. This was superseded by British Standard Specification 490 dated 1943 for the period of emergency caused by the acute rubber shortage resulting from the war. It is probable that a new one will be issued in the course of a year or so. The 1933 Standard refers to three grades of rubber for use in the cover, and to three grades of frictioning compound for binding the plies to each other and the rubber cover to the plies.

The grades for the rubber cover differ both in the ultimate tensile strength and in the percentage elongation. In the case of the frictioning the quality is measured by the force required to separate the plies of duck, or the cover from the adjacent duck, at a given rate,

For test purposes, the upper ply of a strip of belt 1 in wide is separated at one end. To the free end thus formed the specified weight is added, when the rate of separation must not exceed 1 in/min.

The three grades recommended for conveyor belts by the British Standard Specification No. 490 of 1933 are given below :

	Rubber Cover		Friction	
Grade	Tensile Strength, lb/sq.in.	Elongation, %	Adhesion in pounds, ply to ply, 1-in strip at 1 in/min	Adhesion in pounds, cover to ply, 1-in strip at 1 in/min
A	3500	550	26	24
B	2500	450	20	18
C	1500	350	17	15

The strength of the belt in the direction of the warp varies with the weight of the duck. Some average figures are given below, each figure referring to the loading on 1 ply of 1 in width. In addition, there are figures giving the normal thicknesses of the plies for the different qualities of duck, together with the corresponding weights of the ducks in pounds per foot of 1 in width 1 ply thick. The corresponding weight of the rubber covering for a thickness of $\frac{1}{32}$ in is also given.

Wt of Duck, oz :	28	32	36	42
Average breaking strength, pounds per inch per ply	280	300	325	375
Thickness of belts per ply, in inches ...	0·045	0·053	0·056	0·063
Wt of belt in pounds per foot per inch width per ply	0·021	0·024	0·026	0·029
Wt of rubber cover in pounds per foot per 1 in width for $\frac{1}{32}$-in cover	0·018	0·018	0·018	0·018

The thickness and quality of the rubber cover is made to suit the duty of the belt. Generally, for medium-weight material, a cover $\frac{1}{8}$ in thick is sufficient, but heavy abrasive materials require a thicker cover up to $\frac{1}{4}$ in or even more. On the underside of the belt

$\frac{1}{32}$-in cover is sufficient for light duties, and for heavier work $\frac{1}{16}$ in is ample. At the sides of the belt the thickness should be the same as that of the top cover. When the rubber covering for the carrying side is $\frac{1}{16}$ in or over, a frequent practice is to place a layer of open-mesh fabric between the carcase and the cover. When the belt is vulcanized this is thoroughly incorporated in the cover and increases very considerably the bond between cover and carcase.

The majority of belts used are of uniform thickness throughout their width and are known as straight-ply belts. Occasionally a stepped-ply belt is specified, in which the number of plies of duck in the central portion of the belt width is less than that at the sides, one or more of the plies being omitted from the carrying side of the belt. This provides a thicker layer of protective rubber down the middle of the carrying side of the belt where the wear is the greatest, and makes the belt more pliable across its width and better able to conform to the shape of the carrier pulleys which support it.

The improvement in the quality of the rubber forming the protective surface has been so great that the advantage of having an extra thickness of cover over part of the surface is not so marked as formerly and the demand for stepped-ply belts is not so frequent.

Splices—Conveyor belts are most frequently jointed with metal fasteners which are generally satisfactory, but it must be remembered that the metal of which these are made wears more rapidly than the rubber cover of the belt and the joints should therefore be examined at regular intervals and renewed when necessary, in order to avoid serious damage to the belt.

In important belts it is very advantageous to have the ends of the belt spliced and vulcanized in position. Such a joint has double the strength of the metal-fastener connection, and in addition, since there is no break in the continuity of the rubber cover, moisture will not penetrate to the carcase of the belt and cause deterioration. The problem of keeping the belt clean is also greatly simplified, and the possibility of damage due to broken or worn metal fasteners is eliminated. The making of a vulcanized splice calls for considerable care and skill. In the case of works which have a large number of belts, it is well worth while buying a press and training men to use it. In most cases, however, it is better to make arrangements with the belt manufacturers who generally have portable equipment to undertake the work.

The Cord Belt

The cord belt is another type of construction which is suitable for use with hard-worked conveyors, particularly when heavy lumps fall on to the belt at the loading point. The carcase is formed of two parts. Firstly a few plies of heavy cotton duck, sufficiently strong to supply the necessary transverse strength, are arranged at the bottom or pulley side of the belt, then, above this layer, a number of layers of cord fabric are placed. The cords are stouter than those in the cotton duck and are more widely spaced ; there are no cross threads in this part of the belt. The cords, which carry the main part of the tension, are embedded in the rubber, and the structure of the belt withstands the blows at the loading point better than a belt of ordinary construction.

Troughing Idlers

The first belt conveyors were run flat, and were carried by the floor of the conveying trough which gave continuous support to the belt throughout its length. Towards the end of the last century the first troughing idlers were built, and since then considerable improvements in design have been made. The early troughing idler, made for belts up to 30 in width, consisted of three cast-iron pulleys of 5 or 6-in. dia., with their bosses bored to run on short lengths of stout tubes which were suitably carried in cast-iron brackets mounted on a common ' idler board.' The bores of the carrying tubes were connected through the supporting brackets, and by means of large grease cups it was possible to lubricate the pulleys of the idlers through holes drilled in the fixed supporting tubes at the required points. The return idler, carrying the bottom or empty strand of the belt, was formed

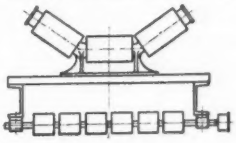

Fig. 2—Arrangement of early type of troughing and return idler

in a similar manner, being made up of three or four cast-iron pulleys mounted on a long cross-tube, which was provided with end brackets for attachment to the conveyor structure. In some cases the return-idler pulleys were fixed on to the shaft which was mounted in suitable bearings carried by the structure. A typical arrangement of an early troughing and return-idler set-up is shown in Fig. 2.

The troughing idler was generally built with one horizontal centre pulley, and two outer inclined pulleys. In very early designs the slope of these outer pulleys was very considerable—up to 45°—in order to obtain the maximum carrying capacity. Unfortunately, with this arrangement of pulleys the belt did not readily conform to the shape of the idler, and though in this respect a stepped-ply belt was better than a straight-ply belt, the empty belt was usually carried on the two inclined rollers only, and heavy loads caused longitudinal lines of weakness in line with the edges of the horizontal centre pulley.

With greater experience it was found that the carrying capacity of the belt was not dependent upon the shape of the idler, but rather upon the shape taken up by the belt in the unsupported position between the troughing idlers. When the loaded belt passed over and beyond an idler it tended to flatten out, and the material upon it spread towards the edges of the belt. On approaching the next troughing idler the belt was constrained to take up the shape of the idler, thus forcing the material back again towards the centre of the belt. This constant movement of the material on the belt caused unnecessary wear of

both the surface and the carcase, the cross-threads of which were subjected to continual flexing. As time went on, the slope of the outer pulleys was gradually decreased, in order to get a troughing idler contour which would conform as closely as possible to the natural shape of the belt. For wider belts five-pulley idlers were used to attain this end.

Bearings

At the beginning of the present century the use of ball and roller bearings was very considerably extended, and the manufacturers of belt-conveyor equipment were quick to recognize the advantages of this form of bearing. Although the reduced friction of ball or roller bearings may, in the case of long horizontal conveyors, enable savings in the cost of belt to be made, this advantage is not so great in cases where the major part of the power is required to raise material from one level to another. Undoubtedly the great advantage of the ball- or roller-bearing idler lies in the fact that the cost of maintenance is so much less. A well-made troughing idler, fitted with ball or roller bearings, has its pulleys balanced better than the journal-bearing type with cast-iron pulleys, and it is therefore possible to run the belt at a somewhat higher speed. With modern methods of manufacture the ball- or roller-bearing idler can be produced almost as cheaply as the journal-bearing type.

Thus the general trend in the design of idler equipment has been to reduce the slope of the side pulleys in successive stages until the comparatively low angle of 20° has been reached by some American manufacturers, this angle also being adopted as the German standard. The tendency is to use three-pulley idlers for all widths of belt up to 60 in. This simplifies the arrangement of the idler, and as there are only six bearings instead of the ten in a five-pulley idler the problem of lubrication is much easier. Some engineers, however, still prefer the five-pulley idler for the wider belts of say 42 in and over, but even with these the slope of the outside pulleys has been reduced to improve the working, and although angles of 15° and 30° for the inner and outer inclined pulleys are still very common, they will probably be reduced in the future—some five-pulley idlers are now being made with angles of $11\frac{1}{4}°$ and $22\frac{1}{2}°$. Another aspect which has contributed to the gradual decreasing of the slope of the side pulleys is the part played by the central horizontal pulley in guiding the belt. If the belt has good contact with this pulley, it will run straight, provided the conveyor is lined-up properly and is centrally loaded. With good contact on the horizontal pulley, it will run straight when empty. This was a difficult matter with the early idlers which had considerable slope to the side pulleys, and led to the use of side guide idlers to keep the belt from running too far off the centre, but at the expense of excessive wear on the sides of the belt. The need for these guide idlers diminished with the angle of slope of the side pulleys. A device frequently adopted in order to improve the contact between the belt and the horizontal pulley is to tilt the idlers forward in the direction of belt travel. With side pulleys at 20° there is no need for any such provision to make the belt run straight, except perhaps as a safety measure when starting up a new installation where the lining-up may be inaccurate. In such a case, when the lining-up has been properly attended to, the guide idlers should be removed.

There are many forms of ball and roller troughing-idlers for use in different circumstances. Most makers fitted ball or roller bearings to cast-iron pulleys of the type used with journal-bearing idlers, but later practice has been to build up the idler pulleys from sections of steel tube, fitted at the ends with cast-iron or pressed-steel end-covers which are bored to take the outer race of the bearing. The inner race is mounted on a through-tube or shaft, carried in brackets mounted on the idler board or its equivalent. Where the material to be carried would cause steel pulleys to corrode, it is usual to supply pulleys in cast iron.

The usual method of mounting the idler pulleys on the ball or roller bearings is to have the outer race fixed in the pulley member, and rotating about the inner race, which is fixed to the non-rotating spindle. Strictly speaking, the action of the bearing is not so favourable as it would be were the outer race stationary and the inner race revolving. There are, however, constructional reasons which make the usual mounting with the rotating outer race preferable. Thus the accurate alignment of the two bearings associated with any one pulley is a simple matter depending only on the operations in the machine shop. It is not affected by any slight distortion which may occur in the supporting member or idler board. With the live-shaft arrangement, the bearings in which each pulley revolves must be accurately mounted upon the idler board or they must be so arranged that there is some measure of self-adjustment.

The load which an idler has to carry depends upon the weight of the material being moved, and so manufacturers usually have two ranges of idler sets, one for light- or medium-weight materials, and a more strongly built one for heavy ores and such material. Generally the load on an idler is fairly constant and subject to a normal amount of shock only, but very robust idlers are necessary at the point of feed where they are subjected to very severe shocks, especially when a heavy ore containing large lumps is involved. The greater part of the wear on the belt itself occurs at the loading point, and it is here that a number of special troughing idlers should be placed to protect the belt from the impact of heavy material. These special troughing idlers are of a heavy type and have a thick layer of rubber from $\frac{3}{4}$ in to 1 in thick over the face of the pulley. Several should be provided at the feed point where they should be spaced fairly closely, say, 15 in to 18 in apart. At the same point an elastic cushion of rubber packing may be placed between the idler board and the conveyor structure. One manufacturer provides a special rubber-disc idler in which the separate pulleys are made up of a number of thick rubber discs, and others market special idlers made up of steel discs with cushion tyres. A special idler of the latter type, in which the tyres can be inflated to any desired pressure, has been produced.

The flat idlers supporting the return side of the belt are also arranged with ball or roller bearings. Instead of a number of separate pulleys, as was usual with the original journal-type idler, a steel tube is used, generally 2 or 3 in wider than the belt. The tube has end caps, either of cast iron or mild steel, which are prepared to receive the outer race of the bearings; the inner race is mounted on a stationary through-shaft. Occasionally the return-idler pulley is fixed to the shaft which rotates in ball or roller bearings attached to the structure. Return idlers are not usually designed to take excessive weight, and care must be exercised not to load them unduly by placing them at a point where the belt changes direction. A bend pulley should be used at such points.

Lubrication

An important point in the design of idlers is the method adopted to lubricate the bearings. Although once greased these should run for several months without attention, the occasional application of grease serves to keep the seals filled and to prevent the entry of dirt and moisture. In some designs where the ball or roller races are mounted on hollow tubes of considerable size, it is feasible to lubricate the whole of the pulleys of an idler set from the end of the tube, as was done with the journal-bearing idler. If the passageway is too small, however, it is better to arrange separate lubrication to each bearing or pair of bearings. This is advisable with the wider belts in any case. Copper grease-pipes can be arranged to run from the side of the belt to the various brackets supporting the pulleys, and grease can be applied to the bearings by means of a grease gun.

DESIGN OF BELT CONVEYORS

In considering the design of a belt conveyor for any duty, conditions may require the belt to be sloped in places. Care must be taken that the angle of the belt does not exceed a permissible amount, varying with the class of material to be carried. Neglect of this precaution may lead to drastic alterations. Naturally, the maximum angle is less for sized material than for mixed, and that for fine material is the highest. If the feed to the belt is not uniform, it is desirable, when handling sized or mixed material, to arrange the conveyor with a curve so that the slope at the feed point is less than the maximum permissible angle. Thus the load is picked up by the belt in a shorter length, and wear on the belt is lessened. Permissible angles of slope for some materials are given below:

	Sized	Mixed	Fines
Coke	17°	18°	20°
Coal	16–18°	18°	20–22°
Iron ore	18°	18–20°	20–22°

When a belt is fed on a slope it cannot take such a full load as it could if loaded on the level, and due allowance should be made. A good rule is to diminish the normal rated capacity of the belt by $1\frac{2}{3}\%$ for every degree of slope. So also, since the work of the belt at the loading point is more arduous with a sloped than with a level belt, it is desirable to run the belt at a slower speed than would be allowed for a level belt.

Speed

The maximum speed of belts depends entirely on factors such as the nature of the material carried, the means of feeding it to the belt, and to a certain extent, on the width of the belt. Obviously, when moving materials such as sized coal the speed should not be so great that excessive breakage occurs. Narrow belts cannot be run at such high speeds as wider ones, as the liability of spillage is greater, and in addition, the guiding action of the narrow centre pulley of the troughing idler decreases at high speeds.

When handling non-abrasive fine material on a belt, the speed should not exceed 600 ft/min for belts 36 in wide and over. For belts under 36 in wide the maximum allowable speed may be taken as $12{\cdot}5\ (W + 12)$ ft/min, where W is the width of the belt in inches.

For abrasive material, to reduce wear on the belt, the speed should be lower. Thus for fine abrasive material the speed should not exceed 85% of the above figure, and for lumpy abrasive material, 70%. If the width of a conveyor for a particular requirement is fixed by considerations other than capacity, for instance by the size of the maximum lump, or by the necessary number of plies in the belt, it is good practice to run the belt at as low a speed as possible in order to reduce wear on the belt. To ensure a good discharge, however, the speed should not be lower than 150 ft/min.

Capacity

In carrying fine material, there is no difficulty in loading the belt to capacity by means of a chute of normal width, which may be taken as two-thirds that of the belt. If the material contains lumps as well as fines, the chute must be sufficiently wide to pass the largest lump together with other sizes. If the material is sized, the chute should be still wider, and naturally the corresponding width of belt will be greater. Cases may occur, when lump material is being handled, in which the necessary width of belt is governed not by the required hourly capacity but by the size of the maximum lump. Manufacturers publish tables stating the size of the maximum lump which a belt of given width will carry when the material is either sized or unsized. The following expression may be used for this purpose. For sized material of which the maximum lump is C-in cube, the minimum width of belt in inches may be taken as $W = 9\sqrt[3]{C^2}$. If the material is unsized, say, 90% being under the maximum size C-in cube and 75% under half that size, a smaller belt may be used. Thus for unsized material $W = 6{\cdot}25\sqrt[3]{C^2}$.

Manufacturers also publish tables of the capacities that they recommend for belts of various widths, running at different speeds, and carrying materials which vary in their specific weight. As a basis for these tables an expression is used which relates the carrying capacity of the belt in cu.ft./hr to the width of the belt, and which takes the form: Capacity of belt in cu.ft./hr at a speed of 100 ft/min is equal to

nW^2, where W is the width of the belt in inches and n a coefficient, usually between 3·2 and 3·75. For such an expression to apply generally to all widths of belt, the area of the cross-section of the material on the belt must also be proportioned to W^2. Thus if the cross-section of the belt be drawn out as in Fig. 3 and the area of the material expressed as a multiple of W^2, it will be seen that if the relationship holds for all values of W, the figure can be considered as applying to any value of W by adopting a suitable scale. It follows that the length of the free portion at the edge of the belt would be proportional to the width of the belt. If, for instance, with a 12-in belt the distance a–b is $1\frac{1}{2}$ in, it would be $7\frac{1}{2}$ in on a 60-in belt. Actually, when a 60-in belt is well loaded the free portion corresponding to a–b in Fig. 3 will be 4 in. Thus capacities based on the expression nW^2 cu.ft./hr at 100 ft/min will be correct for only one value of W. For lower values the calculated capacity will be too high, and for higher values the result given will be below the actual carrying capacity of the belt.

The length of the uncovered portion at the edge of a belt W in wide may be taken as $0{\cdot}05W + 1$ in. Thus the effective width of a belt W in wide will be $0{\cdot}9W - 2$ in. If it is now assumed that the capacity

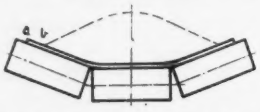

Fig. 3—Load upon cross-section of belt

of the belt is proportional to the square of the effective width, then the capacity of a belt of width W in, travelling at 100 ft/min, will be $K_1(0{\cdot}9W - 2)^2$, or $K_2(W - 2{\cdot}2)^2$ where $K_2 = 0{\cdot}81K_1$. If this expression is compared with one of the ordinary form, which by experience is known to be satisfactory for a particular value of W, an example being, capacity $= 3{\cdot}6W^2$ when $W = 36$ in, it is obvious that $K_2(36 - 2{\cdot}2)^2$ must be equal to $3{\cdot}6 \times 36^2$ which gives $K_2 = 4{\cdot}08$. The following expression is thus arrived at to give the capacity of a belt in cu.ft./hr when operating at a speed of 100 ft/min :

$$\text{Capacity} = 4{\cdot}08\,(W - 2{\cdot}2)^2 \text{ cu.ft./hr}$$

Table I

CAPACITY OF CONVEYORS, CU.FT./HR AT 100 FT/MIN

W	$3{\cdot}2\,W^2$	$3{\cdot}5\,W^2$	$\left(3+\frac{W}{60}\right)W^2$	R	$4(W-2)^2$
12	461	504	461	...	400
14	627	686	634	615	576
16	819	896	836	833	784
18	1037	1134	1069	1077	1024
20	1280	1400	1333	1333	1296
24	1843	2016	1958	2000	1936
30	2880	3150	3150	3158	3136
36	4147	4536	4666	4706	4624
42	5645	6174	6527	6667	6400
48	7373	8064	8755	9231	8264
54	9331	10206	11372	11852	10816
60	11520	12600	14400	14815	13456

This expression is not suitable for easy application, but it can be modified without any serious error by omitting the fractional parts. Thus capacity of belt in cu.ft./hr when running at 100 ft/min $= 4(W - 2)^2$. This is a very simple expression which can be used with confidence.

Table I gives the capacities of belts of various widths in cu.ft./hr at 100 ft/min according to this relationship. For purposes of comparison other capacity figures are also given. Some manufacturers recommend capacity figures according to the form Capacity $= nW^2$ where the value of the coefficient n increases with W. In one of the best known the value of the coefficient n is equal to $3 + W/60$. Capacities in accordance with this formula are also given in Table I.

The capacity as given by the expression $V = 4(W - 2)^2$ is suitable for a well-loaded belt with good feeding arrangement. The maximum capacity would be from 25 to 30% greater, but for design purposes this extra must be considered as a safety margin, to take care of any temporary increase in the rate of flow which may possibly occur in practice.

The maximum capacity which a belt can carry under ordinary conditions may be obtained from the expressions given above for the capacity at 100 ft/min, and the speed relationship. Thus if M is the maximum belt capacity in cubic feet/hour:

For belts up to 36 in wide,

$$M = (W - 2)^2\,(W + 12)\,2$$

For belts over 36 in wide,

$$M = 24(W - 2)^2$$

Further, if the material weighs p lb/cu.ft., and T is the maximum tonnage which can be carried in an hour, it will be seen that :

$$\text{Maximum tonnage per hr } T = \frac{Mp}{2240} \text{ or } \frac{T}{p} = \frac{M}{2240}$$

The value of T/p has been calculated for various widths of belt W and the results are given in Table II. The upper and lower limits of the size of maximum lump carried are also given, in accordance with the recommendations made earlier. Note that the expressions above give the maximum capacities when handling non-abrasive fine material, and that owing to the desirable lowering of speed these capacities should be reduced to 85% when the material is fine and abrasive, and to 70% for lumpy and abrasive material.

Strength

The strength of belt required depends generally upon the power to be transmitted by the belt, which in turn depends upon the layout concerned, the length of the belt, and the vertical lift if any. There are, however, other conditions to be observed, which may result in the thickness of a belt being selected for reasons other than its strength for power transmission. One has already been mentioned in connection with the width of belt necessary to carry materials of different sizes. A second must be considered.

For the empty belt to run true, it must make good contact with the horizontal pulley of the troughing idler. If the belt is too thick in comparison with its

Table II

VALUES OF THE RATIO T/p FOR VARIOUS WIDTHS OF BELT

W	Maximum Lump, inch cube		Non-Abrasive Fines		Fine Abrasive		Lump Abrasive	
	Sized	Unsized	Speed, ft/min	T/p	Speed, ft/min	T/p	Speed, ft/min	T/p
12	1·6	2·7	300	0·536	255	0·455	210	0·375
14	1·9	3·4	325	0·836	276	0·710	227	0·585
16	2·4	4·1	350	1·225	297	1·041	245	0·858
18	2·8	4·9	375	1·714	319	1·457	262	1·200
20	3·3	5·7	400	2·314	340	1·967	280	1·620
24	4·4	7·5	450	3·889	382	3·306	315	2·722
30	6·1	10·5	525	7·350	446	6·247	367	5·145
36	8·0	13·8	600	12·39	510	10·53	420	8·67
42	10·1	17·4	600	17·15	510	14·57	420	12·00
48	12·3	21·3	600	22·68	510	19·28	420	15·88
54	14·7	25·4	600	28·97	510	24·62	420	20·28
60	17·2	29·8	600	36·04	510	30·63	420	25·23

width this contact will not be obtained. There is, therefore, an upper limit to the number of plies which can be used, and this limit will depend upon the width of the belt W, and the weight of the duck.

For 28-oz cotton duck the maximum number of plies may be taken as $W/5 + 2$ for belts from 12 in. to 24 in wide, and $W/5 + 3$ for belts from 30 to 60 in. wide.

For 32-oz duck the number of plies may be less, $W/5 + 1$ for belts up to 24 in, and $W/5 + 2$ for those from 30 to 60 in. wide.

For 36-oz duck, which might be used with belts from 18 in to 24 in wide, the maximum thickness may be as for 32-oz, *viz.*, $W/5 + 1$ plies. Wider belts of 30 in and over, made of 36-oz or 42-oz duck, should not exceed $W/5 + 1$ plies in thickness.

Too thin a belt will tend to flatten out considerably in the spaces between the supporting idlers and cause wear on the surface, owing to the repeated movement of the material. Furthermore, there is a danger of excessive bending of the belt at the edges of the pulleys, in line with which it is possible that the weft threads of the duck will ultimately fail; the belt may even become wedged in the space between adjacent pulleys. The minimum number of plies depends on the type of material carried, the width of the belt and the weight of the duck. Three classes of material will be considered:

A—Fine coal, sand, etc.

B—Lump coal or stone and fine ores

C—Coarse ores and other heavy material.

The following expressions are suggested, to give the minimum numbers of plies it is desirable to use:

Class A material, 32-oz or 36-oz duck—The minimum plies should be $W/12 + 2$. For 28-oz duck one more ply is desirable. Thus for Class *A* material, 28-oz duck, minimum plies $W/12 + 3$. In no case should the number of plies be less than four.

Class B material, 32-oz or 36-oz duck—The minimum plies should be $W/12 + 3$, the corresponding figure for 28-oz duck being $W/12 + 4$, and for 42-oz duck $W/12 + 2$. It should be noted that 36-oz and 42-oz ducks should not be used with belts under 24 in and 30 in wide respectively.

Class C materials, 32-oz duck or over—For 32-oz and 36-oz duck the minimum plies should be $W/12 + 4$, with one less, $W/12 + 3$, for the 42-oz duck. 32-oz duck should not be used for widths less than 18 in, 36-oz for widths less than 24 in, or 42-oz for widths under 30 in.

Power Requirements

Consideration of the power required to transport the material and moving parts of the equipment is necessary, not only to provide sufficient power, but also to ensure that the strength of the belt is adequate for the duty, and that an efficient arrangement is designed for the drive pulley. Many formulae have been devised for this purpose, and a very useful form, easy to apply, is given below. In the following:

W = Width of belt (inches)
T = Capacity (tons/hour)
S = Speed (ft/min)
L = Length between centres (feet)
H = Vertical lift (feet)

It is convenient to consider the horse-power requirement as made up of three parts. Firstly, the power to drive the empty belt at the required speed. Secondly, the power to convey the material over a distance (assumed horizontal), equal to the length between centres. Thirdly, the horse-power necessary to raise the material through the vertical height concerned. The formulae recommended are:

(i) Horse-power to drive empty belt

$$0{\cdot}0125\,(W - 6)\,\frac{L}{100} \times \frac{S}{100}$$

(ii) Horse-power to convey material

$$0{\cdot}275\,\frac{T}{100} \times \frac{L}{100}$$

(iii) Horse-power to raise material

$$11{\cdot}36\,\frac{T}{100} \times \frac{H}{100}$$

The sum of the three powers thus found should be multiplied by a length factor $1 + 20/L$ in order to

provide a margin for the end pulleys, etc., the effect of which is more noticeable in short, than in long, conveyors.

These formulae are for use with ball- or roller-bearing idlers; if journal bearings are used, the powers under (i) and (ii) should be multiplied by three.

The horse-power as obtained above is that applied by the driving pulley to the belt and is referred to as the belt horse-power. The motor horse-power will be greater, owing to the reduction gears between motor-

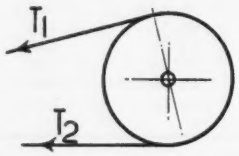

Fig. 4—Horse-power pull on belt at drive pulley, showing tensions at tight and slack sides of belt

shaft and drive pulley. An addition of 5% should be made for each chain or machine-cut gear reduction.

The belt horse-power is used to obtain the horse-power pull, which is equal to the difference in tensions of the belt at the points where it passes on and off the drive pulley. Referring to Fig. 4 :

$$\text{Horse-power pull} = T_1 - T_2 = \frac{\text{Belt horse-power} \times 33{,}000}{S}$$

Here T_1 and T_2 are the tensions at the tight and slack sides of the belt ; it is the maximum tension T_1 which is of importance. Text-books show that the ratio of T_1 to T_2, and consequently the ratio of T_1 to $T_1 - T_2$, is dependent only on the angle of contact of belt and pulley and the coefficient of friction. Generally it is sufficient to consider only two coefficients of friction, one, 0·25, being used for the rubber belt on the bare metal pulley, and the other, 0·35, for the belt on a lagged pulley. The true relationship between the maximum tension T_1 and the horse-power pull $T_1 - T_2$ is not convenient for quick use, and the ratios are conveniently tabulated in degrees for various values of the arc of contact. A simplified relation for bare pulleys is given below, for values of A from 135° to 600° :

$$\frac{T_1}{T_1 - T_2} = 0{\cdot}75 + \frac{200}{A},$$

where A is the angle of contact between belt and pulley.

For lagged pulleys, the ratio will be smaller ; thus for values of A between 135° and 300° take 80%, from 300° to 400° take 85%, from 400° to 500° take 90%. For higher values of A the same value may be used for bare or lagged pulleys.

Working Stresses

When considering the allowable working stress per inch per ply of belt it is necessary to account for the stretching of the belt at various points in the run, especially at the driving pulley where the working tension in the belt is greatest. The elasticity of rubber belts, when stressed to the tensions usually worked to in practice, is practically constant and amounts to 0·15% increase in length for each 1% of the ultimate strength of the belt. If the ultimate strength of the belt in pounds/inch/ply is taken as U lb, it will be seen that an elongation of 1% in the length of a section of the belt corresponds to a load of $6{\cdot}67/100\ U$ lb per inch width per ply.

If a belt consisting of n plies of duck, each t in thick, is passed round a pulley of d-in dia, the outer ply will be stretched more than the inside ply by approximately $2nt/d \times 100\%$. It will be stretched $nt/d \times 100\%$ more than the central ply which may be assumed to be carrying the average tension per in/ply required for the drive. It follows that the outer ply will be stressed more than the centre ply by an amount $6{\cdot}67ntU/d$ or $6{\cdot}67tU/R$, where R is the number of inches of diameter of the pulley for each ply of the belt. If the average tension in the belt at the pulley is P lb/in/ply, the maximum load in the outer ply will be $P + 6{\cdot}67tU/R$ lb/in.

If at a drive pulley a flexural stress equal to one half of the average drive tension is permissible, it will be seen that :

$$\frac{20tU}{R} = \frac{U}{\text{Factor of Safety}},$$

$$\text{or, } R = \text{Factor of Safety} \times 20t.$$

Taking the overall factor of safety as 7·0 then $R = 140t$. If the thickness of duck weighing w oz is taken at $w/800 + 0{\cdot}01$, the value of R for 28, 32, 36, and 42-oz ducks will be 6·3, 7·0, 7·7, and 8·8 respectively.

The actual maximum stress allowed for the combined stresses in this case is $U/7$, of which $2U/21$ is allowed for the horse-power tension. For 28-oz duck at 280-lb ultimate breaking strength, this amounts to 27 lb/in/ply.

Size of Pulleys

As a belt passes around a pulley the different plies stretch to different degrees, according to their distance from the pulley shaft, and the frictioning which binds the various plies together must be sufficiently resilient to allow the slight stretching necessary without failure. It is important, therefore, never to use pulleys too small in diameter, particularly at such places as the drive pulley where the belt may be under considerable tension.

Under ordinary conditions, where belts are in contact with a considerable portion of the pulley, the ratio of the diameter in inches to the number of plies of belt should not be less than 6·0, 7·0, 7·5, and 8·5 for driving pulleys, when using ducks weighing 28, 32, 36, and 42 oz respectively.

For other pulleys, carrying a belt at considerable tension, as at a tripper or at the head of a conveyor with tandem drive, 85% of the above figures may be taken. For tail and take-up pulleys 75%, and for snub and bend pulleys 55% of the figures should be ample, unless they are placed at points where the

belt is under considerable tension, when they should be at least 70% of the size of the drive pulleys.

The figures may be reduced if the belt size has been fixed by considerations other than the power requirement.

Manufacturers usually recommend a working stress per inch per ply of belt, which is used to calculate the necessary thickness of belt from the horse-power tension alone, the figures being sufficiently low to allow some cover for flexural stresses. Thus for 28, 32, 36, and 42-oz ducks, unit belt stresses of 24, 27, 30, and 38 lb/in/ply are recommended. When using these figures the pulley sizes should not be less than those stated above. In certain cases the size of the pulleys may be increased when the working stresses as recommended by the manufacturer could be increased by 10% or more.

In connection with unequal stretching, which occurs in different parts of the belt at various points of the run, the flattening out of the belt from a troughing idler to a flat pulley may be considered.

Two cases are indicated in Fig. 5. The pulley is either located tangentially to the central idler pulleys, or raised above slightly so that the edges and centre of the belt are stretched to the same amount. In both cases the percentage increase of length of the stretched portion over the unstretched is approximately $h^2/2L^2 \times 100\%$, and the corresponding extra stress due to the stretching will be $6{\cdot}67U \times h^2/L2^2$ lb/in/ply, U being the ultimate strength of the duck. If this increase of stress is limited to $1\frac{1}{2}\%$ of U, it is easily deduced that $L = 15h$ approx. If the vertical distance between the edge of the belt and the centre

Fig. 5—Shape of belt between troughing idler and flat pulley. (*a*) Maximum stretch at edges ; (*b*) Edges and centre line equally stretched

line is $W/9$ in, as indicated in Fig. 6, for the case indicated in Fig. 5 (*a*) L should be at least $1\frac{2}{3}W$, and in Fig. 5 (*b*), L can be half this amount. Thus at points where the average tension in the belt is high, such as the head pulley, tripper pulleys, or bend pulleys, the lay-out should be in accordance with Fig. 5 (*b*). At the tail end, the arrangement of Fig. 5 (*a*) is adopted, as otherwise, when running light, the belt might lift from the idlers at the feed point and be damaged by the chute or skirt boards. Since the tension in the belt is a minimum at the tail end, the extra induced stress is not of great moment. The distance between the pulley and the nearest troughing idler is usually fixed by considerations of load and tension of belt, but the distance should not be less than that given above.

Some manufacturers advise the use of crowned pulleys, allowing $\frac{1}{8}$ in per foot width of pulley, or even more. Figures 7 (*a*), (*b*), (*c*) indicate the theoretical lines taken up by the centre and edge of the belt. The ideal arrangement would be that shown in Fig. 7 (*b*), in which the edges and the centre of the belt are equally stretched. The actual arrangement will be generally as shown in Fig. 7 (*a*) or 7 (*c*), in both of which cases to contact the pulley the edges of the belt would have to be more highly stressed than the centre. This does not happen when the distance L is that usual in practice, so that the belt becomes highly stressed in the centre portion,

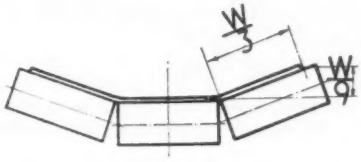

Fig. 6—Elevation of edge of troughed belt

which part transmits most of the driving force, and is under-stressed at the outer parts, where it may not contact the pulley sufficiently to transmit any of the frictional driving force, except over a very reduced angle of contact. Modern idlers should not require the guiding help of crowned pulleys and the practice of those makers who supply only flat-face pulleys with their equipment is sound. Where a change of direction occurs in the carrying strand of a belt it is usual to place a bend pulley, in which case the foregoing considerations apply. With a heavily loaded belt some difficulty may be experienced from the load spreading and spilling due to the comparatively short distance L (Fig. 5). A group of troughing idlers can be arranged to take the place of the bend pulley (Fig. 8). If the radius of the centre line of the belt is R, and that of the edge $R + W/9$, the percentage elongation is $W/9R \times 100$. The extra stress which can be allowed in the edges of the belt depends on the belt tension. If 6·67% of the ultimate strength is allowed,

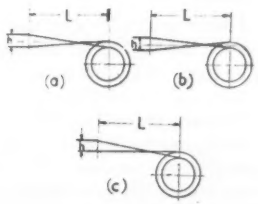

Fig. 7—Shape of belt between troughing idler and crowned pulley. (*a*) Troughed elevation half above and half below crown of pulley ; (*b*) Troughed elevation half above and half below mean pulley diameter ; (*c*) Troughed elevation wholly above crown of pulley

corresponding to a stretch of 1%, it will be seen that $R = 100W/9$. The spacing of the troughing idlers should not be more than usual, and preferably less, as each idler is called upon to carry a radial load due to the curvature of the belt, in addition to the weight of the belt and material. In practice this addition is generally small, but it should be considered in order to ensure that the idler bearings are not overloaded.

Spacing of Idlers

The distance at which the troughing idlers should be spaced depends on the weight of the belt and its load. The greater the load, the closer should be the spacing of the idlers in order that the sag between idlers should not be excessive. Too much sagging not only causes extra wear on the belt, but increases the power required and also tends to make the belt

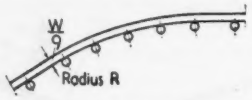

Fig. 8—Arrangement of a group of troughing idlers to take the place of a bend pulley

run crooked. Most makers provide tables giving the spacing recommended for various widths of belt and various weights of material. A reasonable pitch for idlers on belts of width W in, carrying material weighing p lb/cu.ft. is given by the expression: Pitch $= 6{\cdot}5 - W/30 - p/100$ ft. This spacing must be considered as an average figure which has been suited to actual working conditions. Actually, the sag between idlers will depend upon the tension in the belt at the point considered, which may vary considerably with the length and, if any, the inclination of the conveyor. Naturally where the tension is least, at the tail end or feeding point of the conveyor, with a fixed spacing of idlers, the sag will be a maximum. In some cases, where the difference in belt tension at the tail and head ends of the conveyor is considerable, it is worth while varying the pitches of the idlers along the belt, adopting a shorter pitch for a stretch near the tail end, average pitch on the central portion, and a longer pitch towards the head end where the tension is greatest; the spacings should be roughly proportional to the square root of the belt tension.

Grades of Covering

The grade of cover is specified by its tensile strength. The better grades contain more rubber and are more resilient and strong. Since most of the wear on the cover takes place at the loading point, the thickness of cover should vary directly with the speed of the belt and inversely as its length, and also with the class of material to be carried. The cover for a belt handling non-abrasive material may be less than for one handling abrasive or heavy abrasive material, and some difference is desirable to allow for the maximum size of lump in the material.

If it were possible, it would be ideal to have the grade of the frictioning arranged to suit that of the rubber cover, so that when the latter was worn out, the carcase of the belt would be beginning to fail due to the flexing stresses in the frictioning. The loading to be considered in specifying the grade of frictioning depends upon the tension in the belt and its thickness in relation to the size of the pulleys. As with the cover, the grade of frictioning should vary directly as the speed, and inversely as the length of the conveyor, since these factors determine the frequency of the flexing. For belts in regular service and for ratios of S/L of 1 or less, the 17-lb grade of frictioning should be sufficient; for higher values of this ratio, the better grades should be used. This is on the assumption that the pulleys are proportioned according to the recommendations made previously, and that the belt is working at the recommended maximum working stress.

Conveyor belts are subjected to various influences, any of which, with prolonged use, may lead to the ultimate failure of the belt. The cross-threads of the cotton duck are constantly flexing as the belt passes over the idlers, though with the shallow troughing idlers now generally used, and with belts of usual thicknesses, trouble from this source is not frequent. The lengthwise threads have to take not only the conveying force, which varies from the feed point to the driving pulley, but also the extra stresses due to the passing of the belt round the pulleys, which may be considerable if the pulleys used are of insufficient size.

As a rule, the life of a conveyor belt is governed by the wear on the rubber surface, for once this is gone the material of the carcase is unable to withstand the abrasive action of the material carried, and this, together with the deteriorating action of moisture on the unprotected carcase, will quickly render the belt unfit for further service. Practically the whole of the wear on the surface of the belt occurs at the loading point, and though it is impossible to avoid this, it may be minimized by careful arrangement of the chute at the loading point.

Referring to Fig. 9, OA represents the velocity of the material in the chute feeding the belt. The

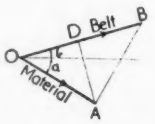

Fig. 9—Velocity diagram at belt feeding point

chute is downwardly inclined at an angle a; OB represents the final velocity of the material and belt which is inclined upwards at angle b. If OA is resolved into components along and perpendicular to the belt, it will be seen that the component AD can have no effect other than pounding on the belt, while the component OD assists by its movement in the same direction as the belt, and therefore requires less power and causes less wear on the surface. It is obvious that the higher the velocity component of the material along the belt, the higher will be the perpendicular or pounding component. The pounding effect is most serious with heavy abrasive material, which may cut the surface of the belt. Idlers must be spaced closely together at the feed point to avoid damage by material wedging under the skirt plates. The pounding component is $OA \sin(a + b)$ where OA is the velocity of the material approaching the belt. The angle a is fixed by the sliding angle of the material on the chute plates, and thus the only means of reducing this pounding component is by making the angle b as small as possible at the feed point, and by reducing OA. The wear on the belt resulting from

equalizing the speed of the material and belt may be minimized by running the belt at no higher speed than is necessary for the duty. If the material contains fines the chute should be arranged with a trough formation, or with a bar screen, so that a layer of fine material is placed on the belt immediately before the lumpy material. The velocity *OA* may in certain cases be reduced by means of bars or chains suspended in front of the chute, care being taken to see that the retarded flow of material does not cause a build-up in the chute. This idea, if carried to the limit, is the equivalent of providing a steady head bunker or hopper at the feed point. With large material the same difficulties would be met as those in bunker outlets required for the certain and uniform discharge of lumpy material—material which is extremely liable to arch over the narrow opening afforded by the conveyor chute. When dealing with a large throughput of heavy abrasive material, provision of a suitable form of feeder, arranged immediately above the main belt, can reduce the pounding action to a minimum.

LOADING CHUTES

Uneven loading of a belt across its width may give rise to difficulty at the loading chute when material is being transferred from one conveyor to another running at an angle to it. The centre of gravity of the load tends to work towards the centre-line of the conveyor idlers and the belt runs out of true. To overcome this, special guiding idlers may be used at about 40- to 50-ft distances. There are several types of these idlers, generally of the same form as the standard troughing idler, but mounted on a central pivot, about which the idler slews when the belt mounts up on one of the side pulleys. The low side of the belt then first meets the inclined side pulley of the special idler which causes the belt to run back to its normal position. Uneven loading can, however, be minimized by careful design of the feed chute, which should be so arranged that the speed of the material on to the second belt is not too great. The more fully the second belt is loaded, the less the liability of uneven loading.

Often, at the discharge end of the belt, it is necessary merely to discharge the material to a bunker or other container. In such cases the speed of flow and the distribution of the material at outlet are of minor importance. The chute should be designed so that the material is led from the belt to the receiver without violent impact on any part of the chute. This is particularly desirable in the case of friable lump material such as coal, undue breakage of which should be avoided, but it is also advisable with any abrasive material which would quickly wear out the chute plates at the areas of contact. The bottom plate of the chute should be arranged to catch the material as soon as it leaves the belt, and should be sloped slightly more than the sliding angle of the material, to ensure a good discharge; it should also be protected with renewable wearing plates, or discharge may be made on to a dirt pocket, in which case some of the material being handled acts as the chute bottom.

REMOVAL OF FINES

Handling fine moist material results in some of the fines adhering to the belt after the discharge point is passed, and unless removed they will cake up on the snub pulley and return idlers and cause the belt to run out of true.

To minimize this possibility a special type of return idler has been developed consisting of a number of comparatively narrow rubber-faced discs mounted on the shaft. Thus there is less area of contact between the belt and the idler face, and any material which is loose falls clear of the idler.

With a belt running say 10 hr daily the amount of material involved in the course of a day's run is very considerable, and without special provision much labour and expense is required to deal with the fines which drop from the underside of the conveyor throughout its length. The best place to remove this fine material is at a point between the discharge and snub pulley, before contact with the latter has caused part to adhere to the pulley and part to be pressed more tightly on to the belt surface. A rubber scraper placed just behind the head pulley will remove the greater part of the adhering fines; it is necessary to arrange this so that it is held to the belt by a weight or spring, and frequent inspection is required, especially if the joint in the belt is made with metal fasteners. For extreme cases, where the material is damp and sticky, the only satisfactory method is to provide a rotating brush driven from the conveyor pulley or by a separate motor. This should rotate against the belt at a speed sufficiently high to throw the material clear of the brush. The ordinary bristle brush does not stand up long to the duty, and the use of a spiral-grooved cylinder of soft rubber is more satisfactory.

The necessity for this auxiliary should be recognized from the start and arrangements made to provide ample room for inspection purposes; adequate means should also be provided for the removal of the fine material.

Accidental damage to the belt may be caused by material from the carrying side of the belt falling on to the return strand, or tools or other bodies finding their way there by accident. Therefore it is always advisable to have the conveyor structure covered in between idlers, so as to form a continuous protection for the return strand of the belt. Should the rubber cover become pierced at any time, due to accident or the heavy pounding of sharp material, the cover should be repaired as soon as possible, either with rubber patches which the makers can supply, or preferably, by the use of a small patch-vulcanizer. Such attention will prevent the entrance of moisture.

STORAGE BUNKERS

In most works, facilities for the storage of material have to be provided, and arrangements are usually necessary both for long and short periods of storage.

Long-period storage arrangements are dependent on the nature of the material concerned and the amount of space available for storage. Some materials deteriorate if exposed to the weather, and it may

therefore be necessary to cover any large store with a roof. Again, as in the case of coal, stock heaps must be limited in height. Generally, for periods of long storage, stock-piles will be formed, and only in special cases where the available space is very limited, or the land very costly, would bunkers be considered for such purposes.

Bunkers are preferable for short-period storage, and have the advantage that the material can be withdrawn from them with comparative ease. For small capacities they may be constructed of steel, but for large capacities reinforced concrete is used. The latter type of bunker requires less maintenance than does a steel one, though care is necessary to ensure that the inside faces and edges at the outlet are soundly constructed. However, modifications which may be desirable to meet the needs of the plant served are not carried out so readily as with a steel bunker, and a further disadvantage is that when no longer required the cost of the structure is a total loss.

Bunkers used for short-period storage play a very important part in most plants, and the efficient running of the works is only ensured if the bunker and its outlets are designed to permit the smooth passage of the material through them. When the lower parts of the bunker are made up of sides sloping in two directions, it must be remembered that the minimum slope at the corner angles should be sufficient for the material to flow down with certainty, as otherwise it will prevent the free flow of the material down the more steeply sloping sides. In the case of steel bunkers the inner surface should be free of rivet heads or any obstructions which might cause the material to hold up until the sliding angle corresponds to that of the material on itself. Some care is necessary in the consideration of the design of the outlet gate. The size of gate is most frequently governed by the size of the material to be passed though it. Thus for fine materials, discharge from the bottom of the bunker can be by horizontal sliding plates which are easily operated by hand. As the size of lump increases, so also must the size of opening, and the effort of operation. With very coarse material, the opening required for free flow of the material becomes so great that not only is it necessary to supply power-driven gates, but a secondary chute or feeder is required to regulate the flow of material. Simple slide gates are very suitable for use in the smaller sizes for fine and freely running materials, but for easier operation, and for use with lump material, the plates should be mounted on rollers, or preferably, arranged to turn about a horizontal axis, the closing gate then being curved to suit the radius of the opening. With horizontal gates on bunkers holding coarse material some difficulty will be experienced in closing the gate against the flow of material, and unless a secondary controlling chute is provided, this type of gate should not be used if it has to be closed while material is still in the bunker. For heavy lumpy material, it is better to arrange the outlet in the side of the bunker, as difficulties due to the arching of the material in front of the opening are more easily dealt with. Side openings to the bunker will generally be fitted with a discharge chute to guide the material to the desired point, and further control of the flow can be conveniently made by means of radial gates in the chute.

When lumpy material from the bunkers has to be moved from the bunker outlet to its destination by means of a conveyor, it is very necessary to provide a suitable feeder between bunker and conveyor to ensure that the belt shall not be overloaded, and at the same time to ensure that the bunker opening shall be sufficiently large to pass the maximum size of material without the possibility of arching before the opening.

When heavy and bulky material is to be loaded from a bunker to a second container through a side opening with a discharge chute and control gate, it is convenient to close the bunker opening by means of a grid-like system of flat bars, so that when necessary the material can be loosened with a rod while the gate is closed, thus minimizing the possible danger to workmen should the material come down with a sudden rush. This is more necessary with hand-operated gates, for with power-operated gates the opening can be made large enough to avoid stoppages.

FUTURE DEVELOPMENTS

The chief points of importance in the make-up of conveyors and the necessary parts have been considered in the above notes. With regard to further developments in belt manufacture, it is not improbable that rayon will be used in the structure of the carcase. As rayon is much stronger than the strands of cotton fibre at present forming the duck, it should be possible to arrange the body of the belt in such a manner that the warp threads are contained in a fewer number of plies than is now necessary, thus minimizing considerably the flexing stresses and making it possible to use smaller pulleys.

With regard to the idlers, a number of manufacturers of ball bearings are now producing sealed bearings which are intended to work continuously without any lubrication, the grease applied at the factory being sufficient for the life of the bearing. The success of such bearings is largely dependent upon the properties of the grease used as lubricant. Some manufacturers are considering the use of oil as the lubricant. Some time will be necessary in order to develop and prove such innovations which would improve the efficiency and reliability of the conveying equipment.

The above paper by Mr. J. Brimelow and the following paper by Mr. T. H. Stayman and Mr. D. R. Brown, together with the paper on "Ore Discharging" by Mr. G. T. Shoosmith, published in the September issue of the *Journal*, will be discussed at the Eighth Meeting of the Iron and Steel Engineers Group, on 24th November, 1948.

Specialized Rolling Stock for Iron and Steel Works

By T. H. Stayman, A.M.Inst.C.E., and D. R. Brown, B.A.Eng.(Cantab).

Part I—TRANSFER CARS, *by T. H. Stayman*

THE modern blast-furnace making 1000 tons/day requires considerable quantities of ore, the quantity varying with the iron content and, in addition, several types and grades of ores are normally used.

The main features of a blast-furnace stocking and charging system should comprise a yard for storing ore, and a series of elevated material bins from beneath which the ore, limestone, and other materials are taken *via* the scale car (*see* Part II) to the furnace-charging skips. The material can be grabbed from the stockyard by an ore bridge and placed in an elevated hopper on the furnace high-line. From there it can be taken to the bins by conveyor or transfer car.

Where only one or two furnaces are to be served this material can be economically handled by a conveyor belt with trippers, but for more than two furnaces this system becomes very complicated and a transfer car should be used.

A transfer car is essentially a self-propelled wagon fitted with side or bottom doors and a compressor of suitable size to supply air for operating the doors and brakes.

Very large quantities of material are handled by the car, causing considerable wear on the hopper, which is the cheapest part of the car, and may lead to the whole car being out of use while repairs are carried out. This means that the use of much expensive equipment in electrics, compressors, etc., is lost while repairing relatively inexpensive parts of the car. To overcome this a car has been considered where the whole hopper can be removed from the underframe and repaired at leisure while a spare hopper is fitted, but the design has not yet been fully developed.

Doors

Careful consideration should be given to the design, construction, and lining of the hopper and doors. The type of doors, *i.e.*, bottom or side discharge, will be determined by the bin arrangement but they should be as large as possible and allow the car to clear itself.

Time per Cycle

The time per cycle is influenced considerably by the charging and discharging time, a typical case being a 50-ton car with a maximum run of 1000 ft. This car is designed to give a top speed of 10 m.p.h. with an acceleration of 1·5 ft/sec/sec, taking 154 sec to travel the full 1000 ft and return. The cycle time is 6 min, thus allowing 3 min 26 sec for charging and discharging.

Types of Bogies

Bogies may be of plate frame, diamond frame, or cast steel. When using plate-frame bogies laminated springs are placed immediately above the axle boxes, while diamond-frame and cast-steel-frame bogies use coil springs placed centrally between the axle boxes. The laminated springs are arranged so that they can be adjusted before or during operation, but the coil springs do not allow for any adjustment.

The maximum wheel load should not exceed 16 tons and track restrictions may impose a lower limit. Cars with a capacity appreciably in excess of 60 tons should be carried on six-wheel bogies, which in English practice are generally of the plate-frame type. To ensure that the middle wheels do not carry an undue proportion of the load the frame is made sufficiently stiff to carry a considerable portion of the load when supported on the outer springs only, and the tension of the centre springs is adjusted so that the outer springs carry more than one-third of the initial load. As the load is applied the outer springs become loaded first and the total load is shared. This method is satisfactory for all normal conditions.

Another method of avoiding unequal proportion of the load is to incorporate roller-bearing axle boxes and equalizing gear between springs. The spring inner eyebolts pull on to rocker arms, pivoting on hardened steel pins in cast-steel brackets bolted to the frame. Increased tension on any one spring is immediately shared with the other two. This method is often found on main-line wagons and loco tenders but is not generally used on transfer cars unless track conditions are bad. Bogie pivots should be large and provided with an oil well fed from an oil box placed near the side of the car for easy filling.

Types of Drive

The axle-hung, nose-suspended traction motor is generally considered to be the simplest and most robust form of drive for this type of car. Straight spur gears are employed, the whole lending itself to ease of maintenance and trouble-free running.

An industrial-type motor, mounted on a platform rigidly suspended from the bogie is not considered good practice, the main disadvantage being in the method of drive between the motor and axle. With

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Parts I, IV, and V are by Mr. Stayman, who is Chief Engineer of the McKee Iron and Steel Division at Messrs. Head, Wrightson and Co., Ltd., Thornaby-on-Tees. Parts II and III are by Mr. Brown, who is Blast-Furnace Engineer at Messrs. Ashmore, Benson, Pease and Co., Ltd., Stockton-on-Tees.

the axle normally sprung there will be relative movement between axle and motor shaft owing to the upward and downward oscillation of the wheels and axles, limited only by the axle boxes in the horn gap. It is of course impossible to use straight spur gears as the gear centres are variable and thus accurate meshing cannot be obtained.

Chain drives in which the slack in the chain allows for variation in centres have been devised, but severe strains are imposed on the chain during starting and under varying-load conditions. In reversing, the shock in taking up the slack in the chain is very severe and tends to increase the maintenance and spares requirements. To avoid such troubles, cases have been known of an industrial-type motor being mounted on a platform supported on axle bearings, making it, in effect, an axle-hung motor.

The traction type of axle-hung motor is far superior under running conditions as it has been designed for the severe duties imposed upon it. Both horizontal and vertical drives with bevel and worm gears are possible but with very much increased cost and maintenance. Three-phase A.C. axle-hung motors have not yet been developed for this type of work. With three-phase operation three wires have to be run along the track or alternatively two wires and one running rail may be used. The first of these schemes becomes difficult if there are any cross-overs or turn-outs on the track and for the second scheme a special transformer is required. A further limitation is the choice of motor, which would have to be a wound-rotor, three-phase induction motor. Speed control is difficult, but for transfer-car work where one speed only is required this type of motor offers practical possibilities. Its physical dimensions would be greater than those of a similar D.C. motor and the question of accommodating the motor arises.

The heavy starting torque required governs the physical size of the machine and difficulties arise in gearing. For heavy-duty work, double reduction gears would be necessary, involving higher initial cost and greater maintenance than the straight spur gears of the normal D.C. machine. On the basis of initial cost, however, for some services A.C. motors would be more economical than the building of a rectifier substation to feed the line. For example, a straight track some 500 ft long with no cross-overs, etc., and a supply of three-phase, 50-cycle, 500-V, with low accelerating rates on the car, would be cheaper with A.C. motors than with D.C. motors and rectifiers, but hard and fast rules cannot be drawn as the A.C. system is a possibility and each proposed A.C. scheme should be considered on its merits.

Track Resistance

The value of track resistance for this type of work depends mainly upon the condition of the running track. The resistance due to the type of bearing (plain or roller), lubrication, weight per axle, speed, wheel diameter, etc., although existing, is relatively small compared with that due to irregularities of the track.

The track resistance varies from about 10 to 40 lb/ton on level track at constant speed ; starting conditions impose an appreciably higher resistance, especially for plain bearings, until the bearing surface becomes lubricated. The resistance is increased considerably if the car is starting on irregular track or against a rail joint. Foreign matter, high rail joints, wide gaps at rail joints, widening of the track gauge, etc., all have very large effects on the average track resistance.

The flexibility of the rails, especially where ballasting is not maintained at the high standard demanded for heavy axle loads, can add appreciably to the track resistance, as in mounting the short inclined plane produced the car does work on the ballast, and this energy is wasted unless the ballast pushes itself back into place.

The impact of the wheel against rail heads dissipates energy in heat into the wheel.

Unless track maintenance is of a very high standard and track irregularities are non-existent, an average track-resistance value for a 100-ton car could not be considered less than 15 lb/ton running laden and 25 lb/ton starting laden, both these figures being increased by 5 lb/ton for the car running empty.

Brakes

Transfer-car brakes are generally operated either by air or by hand, pneumatic brakes working at a pressure of 50 lb/sq.in. or more. The brakes should work on eight wheels and give a total block pressure equal to 80–85% of the tare weight of the car with a brake leverage of 8 to 1. For air brakes a leverage of 8 to 1 should not be exceeded.

Hand brakes are for parking or for emergency use and are usually of the screw type, working through the same levers as the air brakes and connected by a slotted link or pull rod. The total block pressure of hand brakes is about 25% of the tare weight of the car.

A compressor complete with air reservoir should be fitted on the car and should be of ample capacity to operate the doors and brakes ; a typical size for a 50-ton car would have a piston displacement of 30 cu.ft./min.

Current Pick-up

In a third-rail system the supply rail is usually outside the running rail. The Factory Acts do not permit ordinary railway practice in the design of collector systems for steelworks use as the supply rail must be adequately safeguarded against the risk of accidental contact. This means that the supply rail must be covered at the top and on both sides; a convenient method is to allow the shoe to ride on the underside of the rail. Various sections may be used for the supply rail and in some cases specially designed sections are available. A bull-head section mounted on its side has been found to give good results.

A typical transfer-car specification is given in the Appendix.

Appendix

SPECIFICATION OF A TYPICAL TRANSFER CAR

Figure 1 shows an arrangement of a typical 50-ton transfer car.

General—The car shall have a capacity of 830 cu.ft., being capable of carrying 50 tons of ore and suitable for running on standard gauge (4 ft 8½ in). It shall be electrically propelled and capable of a top speed

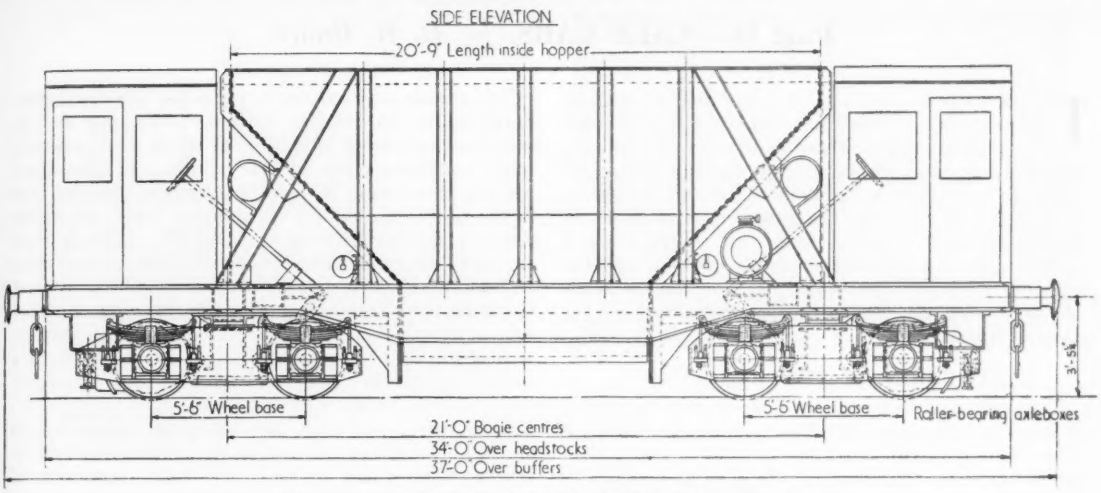

Fig. 1—Arrangement of a typical 50-ton transfer car

of 10 m.p.h. with an acceleration of 1·5 ft/sec/sec. The car shall be loaded from overhead bunkers and discharged through longitudinal bottom doors.

Underframe—The underframe shall be of ample proportion to withstand the loads and impact. It shall be constructed of rolled steel plates and sections, riveted or welded, and designed with a high factor of safety to give low stresses and deflection under full load. Headstocks, bolsters, cross-bars, longitudinals, and diagonals shall be connected by deep knees and gussets.

Body—The body shall be constructed of plates and sections with renewable liner plates of carbon steel. It shall have a capacity as filled of 50 tons of ore at 130 lb/cu.ft., and side and end plates shall be at such an angle as will allow the car to clear itself without poking.

Cabs—A cab shall be erected at each end of the car, constructed of angles and plates. Each cab shall be fitted with two doors and five windows placed three in front and one at each side. A ventilator shall be placed in a suitable position. Each cab shall be arranged for complete control of all operations.

Bogies—The bogies shall be of the plate-frame type constructed of rolled steel sections and supported by laminated springs on roller-bearing axle boxes. Wheels and axles shall be in accordance with Railway Clearing House standard where applicable. The wheels shall be solid forged, or forged or cast-steel centres with rolled steel tyres. Pivots shall be of cast steel with phosphor bronze liners arranged with an oil well for lubrication.

Pneumatically operated sand boxes shall be fitted at each driving wheel.

Doors and Door Gear—The doors shall be of ample size, constructed of steel plates and sections, and provided with renewable carbon-steel liner plates. They shall be pneumatically operated through toggle gear which shall be self-locking. The equipment shall include one motor-driven air-compressor, two double-acting cylinders, and two reservoirs and all necessary valves, cocks, gauges, etc.

Brakes—Brakes shall be pneumatically operated and arranged to operate on four wheels of each bogie. They shall develop a block pressure of 80% of the tare weight with a cylinder pressure of 50 lb/sq.in.

An emergency or parking brake, hand-wheel operated, shall be arranged to work through the same levers as the air brake, being connected by means of a slotted link or pull rod.

Buffing and Drawgear—Buffing and drawgear shall be to Railway Clearing House standard, the buffers being of forged steel, self-contained, with 18-in dia heads.

Motors—The motors shall be axle-hung, nose-suspended, totally enclosed, weatherproof, series-wound, traction types, fitted with commutating poles, and shall be supplied complete with the necessary gearing enclosed in a suitable gear case. Two motors shall be fitted, one on the outboard axle of each bogie. Design shall be in accordance with B.S.S. 173—1941.

Control Gear—The car shall be fitted with a master controller and switching equipment in each of the two cabs. One cab shall house the contactor panel and necessary resistances together with the circuit breakers and ancillary gear. The master controllers shall be interlocked so that only one may be in use at any time and each must be capable of controlling motion in either direction. Control gear shall be of the railway type with a 'dead man's handle' feature so that in the event of the driver releasing his handle the power is cut off and the pneumatic brakes are applied.

The motors shall be series-parallel controlled and the necessary steps of resistance notching shall be provided. No provision need be made for electrical braking other than plugging in case of emergency. Automatic acceleration shall be provided by means of a current-limiting relay.

The contactor panel shall be enclosed in a metal case and the resistances shall be of the unbreakable type, enclosed and adequately ventilated.

Lighting—An interior light and control switch shall be included in each cab. At the front of each cab shall be two headlamps, one clear and one red. They shall be arranged so that the red lamp at one end and the clear lamp at the other end shall be alight at the same time.

Part II—SCALE CARS, *by D. R. Brown*

THE most obvious service that the mechanical age could render to the blast-furnace was to banish the primitive and laborious system of charging by hand-barrows. If the change was not prompted solely on humanitarian grounds, it became necessary as the size of furnaces outgrew the capabilities of hand-charging. It also offered an escape from a system open to slipshod methods of weighing the charge. Thus the mechanical charger and the scale car were amongst the earliest contributions to the modern furnace.

FUNCTION OF THE SCALE CAR

The scale car draws from the bins weighed quantities of the raw materials—ore, limestone, and sometimes coke—and conveys these to the point where they are handled by the charger. With most systems of bucket chargers, the scale car, or larry car, consisted of a self-propelled chassis fitted with a weighing device to which the buckets were in turn transferred for filling. However, as bucket charging is already regarded as obsolete, the present account will deal only with scale cars intended to serve the skip-filled furnace. The field to be covered can be further narrowed down by avoiding the freak arrangements that can occasionally be found, such as telpher cars and single cars serving two or more small furnaces, and by confining attention to the types of car intended to charge furnaces capable of producing 500, 800, and 1200 tons/day. The task which falls to the scale car, even when using a moderately rich burden and when coke is charged without the intervention of the car, will be to handle approximately the following amounts according to the furnace output :

Furnace Output, tons/day	Wt. to be Handled, tons
500	1150
800	1850
1200	2750

If lean ores are to be used, the above tonnages may well be increased by 50%. Such figures are formidable ; for how many industrial cars are called on to load and discharge over 1000 tons/day for 365 days a year? Scale cars are therefore heavy-duty equipment and must not be confused with the many types of light railway equipment used for industrial purposes.

SCALE-CAR CAPACITY

The selection of a scale car for a particular furnace is a matter for proper correlation. The capacity of the car hopper may advisedly be made the same as the volume of the skip tub, which in turn is usually dictated by the desirable depth of the coke layer in the furnace throat. Skip tubs may vary considerably in size, again depending upon the furnace output, but the following figures give a rough indication :

Furnace Output, tons/day	Capacity of Skips, cu. ft.
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800	165
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The reason why skip sizes have not increased proportionately to furnace outputs does not lie in increased speeds of operation so much as in making more continuous use of the equipment, abolishing waiting time as far as possible. A step towards this is the use of a double-compartment scale car which reduces the travelling time when collecting materials from the bins and then allows two consecutive tubs to be filled without delay. If this type of car is used in conjunction with automatic coke charging, the load for every skip can be ready waiting as each skip in turn descends into the pit for filling. Double-compartment scale cars are generally used for furnaces making about 750 tons/day and upwards.

Designers must plan the charging equipment to be capable of filling the furnace at speeds considerably higher than would be the case if the sequence of operations invariably proceeded without intermission ; for a furnace working irregularly may for lengthy periods offer no room for further charges, and then a heavy slip calls for speedy filling until the normal stock level is again recovered. The charging equipment must be regarded as an integrated collection of units, consisting of scale car, skip-hoist, distributor, small bell, and large bell ; each must synchronize with the others and all must operate within the speed that the nature of the equipment can allow. The scale car must be capable of fairly vigorous acceleration and braking, but excesses must be avoided as 50 tons on two axles or 80 tons on four axles cannot be violently accelerated and retarded every minute of the day without undue wear and tear.

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The main principles to be achieved in the design of scale cars are as follows :

(1) The car must be absolutely reliable so as to ensure 100% availability when maintenance is carried out only at prescribed intervals.

(2) The operator must be relieved of all distracting duties, so that he can concentrate on correct performance of his routine and on the accurate weighing of the load.

(3) The opportunity for human error must be removed as far as possible.

(4) Safety.

However reliable a scale car may be, the time must come when repairs are essential and so some degree of spare capacity is needed. This may involve three cars to serve two furnaces, or four to serve three furnaces. An installation of this sort enables the cars to be serviced according to a set routine and it is not asking too much to expect dependability during the duty periods. As an example of availability and reliability, the furnace holding the country's output records has for two years operated with one car owing to circumstances of a temporary nature which, as yet, allow no room for a second car. Chief credit for the performance must be awarded to those who have been responsible for maintaining the vehicle in

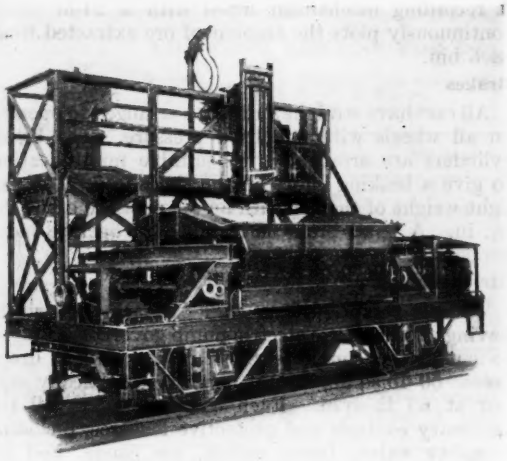

Fig. 2—9-ton scale car. Type *A*

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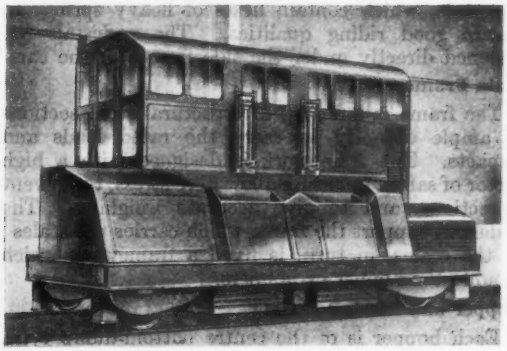


Fig. 3—20-ton enclosed scale car. Type *B*

DESCRIPTION OF VARIOUS TYPES OF SCALE CARS

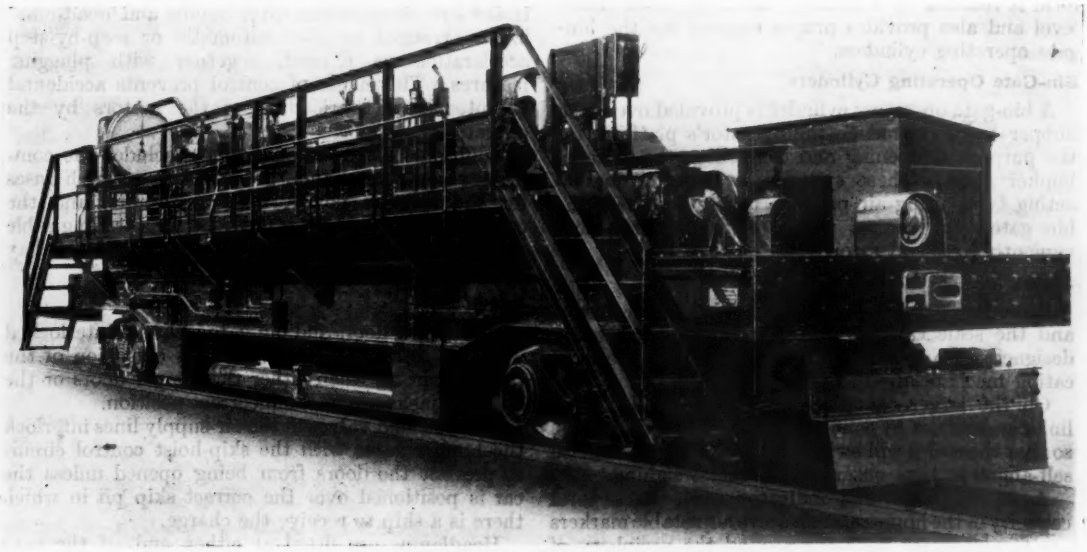
Three types of scale cars are considered: *A* (Fig. 2), suitable for a furnace making 500 tons/day; *B* (Fig. 3), for 800 tons/day; and *C* (Fig. 4), for 1200 tons/day. All three types are electrically propelled and of the bottom-dump type. Cars *A* and *B* have four wheels, and car *C* is mounted on bogie trucks. Heavy pipe lever scales are connected to the weight-indicating mechanism.

The capacities of the three types are as follows:

Car	No. of Hoppers	Level-Full Capacity of Each Hopper, cu. ft.	Max. Load (Each Hopper), tons
A	1	150	9
B	2	165	10
C	2	275	16½

Wheels, Axles, and Springs

The 33-in. dia steel wheels are pressed and keyed on to forged steel axles mounted on roller bearings and enclosed in cast-steel axle boxes. The weight is transferred from the frame to the axles by cast-steel



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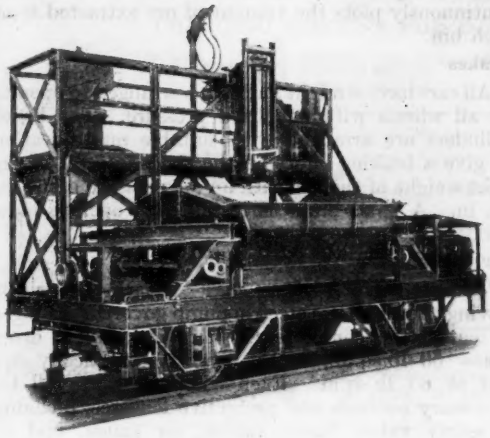

Fig. 2—9-ton scale car. Type *A*

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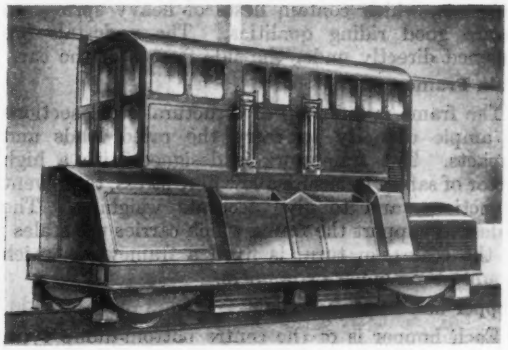

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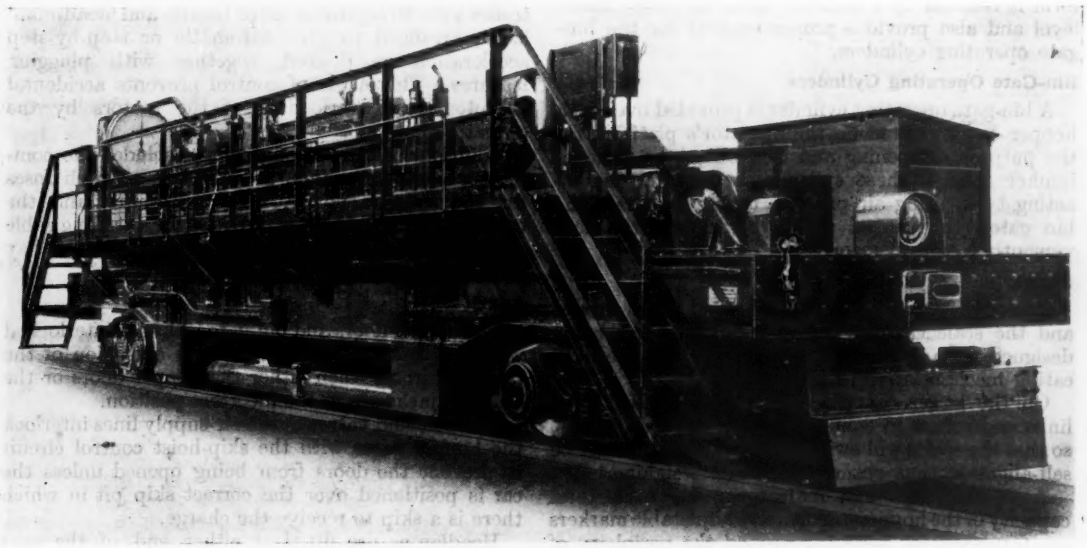


Fig. 4—33-ton scale car. Type *C*

pedestals, which contain nests of heavy springs to ensure good riding qualities. The pedestals are fastened directly under the side frames of the car.

Main Frame

The frame is built up of structural steel sections of ample capacity to carry the rated loads and impacts. The framework is designed with a high factor of safety to ensure a low fibre stress, to prevent deflections, and to give accurate weighing. The pedestals support the frame which carries the scales ; to the scales, in turn, is fixed a framework which mounts the hopper or hoppers.

Hopper and Discharge Gate

Each hopper is of the centre bottom-dump type, discharging between the rails, the gate being opened and closed by a double-acting air cylinder operated by a four-way valve from the operator's platform. Each hopper gate is stiffened by angles and the sloping portion of the hopper, and the inside surfaces of the gates are lined with wearing plates. The action of the gate is such that it drops away from and moves out of the path of the discharge when the hopper is dumping.

Gate-Operating Mechanism

The gate is operated by means of a shaft, at each end of which a single-arm lever is mounted. Steel links connect these levers with the gate. The air cylinder is mounted on the hopper for the operation of the gate by means of a rack and pinion connected to the shaft. A housing is provided over this rack and pinion to exclude ore dust and other foreign matter.

Operator's Platform

The operator's platform is carried on a framework above the hopper or hoppers and extends practically the entire length of the car. Mounted on this platform are the controller, brake wheel, scale dial, air valves, and other operating equipment of the car ; the platform is reached by a stairway from the main frame level and also provides proper support for the bin-gate operating cylinders.

Bin-Gate Operating Cylinders

A bin-gate operating cylinder is provided over each hopper and mounted on the operator's platform for the purpose of opening and controlling the storage-bunker gates. These cylinders are of the double-acting type, using air pressure to lift or release the bin gates, the control valves being mounted conveniently to the operator.

Scales

The scales are designed to withstand rough service and the scale castings are of steel, all parts being designed for a capacity double that which the indicating mechanism will register.

On the larger cars the scales are provided with links and rollers in connection with the knife edges, so that the scales will swing freely at all times and be self-aligning, thus ensuring accurate weighing.

A 24-in. weigh dial indicator reads to a total capacity of the hoppers, and twelve adjustable markers are furnished for fastening around the periphery of the dial to indicate the amount of the various weights. A recording mechanism fitted with a 24-hr chart continuously plots the amount of ore extracted from each bin.

Brakes

All cars have straight air brakes arranged to operate on all wheels with equalized pressure. The brake cylinders are arranged with suitable multiplication to give a braking effort on the shoes of 70% of the light weight of the car with an air pressure of 50 lb/sq. in. A hand brake is installed to operate independently.

Air Compressor

The air compressor on each car is motor-driven, having an automatic electric pump governor arranged to cut-in the compressor when the pressure drops below 50 lb/sq.in., and it again automatically cuts out at 65 lb/sq.in. Each compressor has all the necessary controls and protective features, including a safety valve, fuses, switch, air gauge, and air reservoir.

ELECTRICAL EQUIPMENT

The following details apply to the electrical equipment of the three types of scale car under review :

Motors

The motors for all three types of car are totally enclosed, series-wound, working through single reduction gears :

Car	No. of Motors	Horse-Power of Each Motor	Speed, r.p.m.
A	1	35	520
B	2	45	430
C	2	75	400

Control Gear

One totally enclosed contactor panel, complete with all the necessary contactors and relays, is arranged to give series/parallel control of the motors ; they are protected from damage by hand-reset dash-pot-type magnetic overloads. The contactor panel operates in conjunction with one totally enclosed master controller with straight-line drive handle and 'dead-man' latch, arranged to give automatic or step-by-step acceleration as desired, together with plugging features. This mode of control prevents accidental or intentional overloading of the motors by the operator.

Additional electrical equipment includes the compressor contactor panel, main isolating-switch fuses for the traction power, lighting circuits, and the collector gear ; a socket is provided for a trailing cable which may be used to move the car to a repair bay where no live rails may be available.

Interlocks

The hopper doors and bin-gate jacks are interlocked with the control circuit to prevent operation of the traction circuit when either the hopper doors or the bin-gate jacks are in the incorrect position.

Two solenoid valves in the air-supply lines interlock the hopper gates with the skip-hoist control circuit to prevent the doors from being opened unless the car is positioned over the correct skip pit in which there is a skip to receive the charge.

Headlamps are fitted at either end of the car ; they work in conjunction with auxiliary contacts

Table I

CHARGING ROUTINE FOR DOUBLE-HOPPER SCALE CAR, TYPE *B*

D = Distance from centre-line of bunker to centre-line of skip pit (feet)
N = North of skip pit
S = South of skip pit

	1st Skip (North)				2nd Skip (South)				3rd Skip (North)				4th Skip (South)			
	D	Ore	Wt, cwt	Total Wt, cwt	*D*	Ore	Wt, cwt	Total Wt, cwt	*D*	Ore	Wt, cwt	Total Wt, cwt	*D*	Ore	Wt, cwt	Total Wt, cwt
1st Round	N135	Spathic	48	96	N123	Spathic	48	110	N25	Zaccar	48	96	N25	Zaccar	48	109
	N77	Oxford	48		N77	Oxford	48		N77	Oxford	48		N77	Oxford	48	
					S50	B. slag	14						S30	L'stone	13	
2nd Round	N25	Zaccar	48	96	N25	Zaccar	48	110	N25	Zaccar	48	96	N25	Zaccar	48	109
	N54	Oxford	48		N54	Oxford	48		N54	Oxford	48		N54	Oxford	48	
					S50	B. slag	14						S30	L'stone	13	
3rd Round	N25	Zaccar	48	96	N25	Zaccar	48	110	N25	Zaccar	96	96	S125	Zaccar	96	109
	N54	Oxford	48		N77	Oxford	48						S30	L'stone	13	
					S50	B. slag	14									
4th Round	S73	M.S. Orne	96	96	S96	Kiruna	96	110	S96	Kiruna	96	96	S96	Kiruna	96	109
					S50	B. slag	14						S30	L'stone	13	

on the master controller and throw a red warning beam in the direction of the car's motion.

SCALE-CAR PERFORMANCE

Some conception of the duty demanded of a scale car during normal operations can be derived from the following figures and from Table I:

Type A—Single-Hopper Scale Car

Furnace make (at time of test)	650/700 tons/day
Rate of descent of burden	10 ft/hr
Length of bin system north of skip pit	90 ft
Length of bin system south of skip pit	150 ft
Charging routine during test	OOOCCC Dump OOCC Dump
Coke charge per skip	3500 lb automatically delivered

Skip 1 : (i) Collect 5000 lb Grangesberg from bin 75 ft south of skip pit
(ii) Collect 3200 lb Naverette from bin 45 ft north of skip pit
Average time required = 2 min 25 sec

Skip 2 : (i) Collect 6250 lb sinter from bin 150 ft south of skip pit
(ii) Collect 2950 lb limestone from bin 30 ft north of skip pit
Average time required = 2 min

Skip 3 : (i) Collect 3600 lb Ouenza from bin 105 ft south of skip pit
(ii) Collect 5700 lb Wabana from bin 45 ft south of skip pit
Average time required = 2 min

Skips 4, 5, and 6 : Coke automatically charged
Average wait before dumping large bell at 10-ft stockline = 3 min

Skip 7 : Collect Grangesberg and Naverette as for skip 1

Skip 8 : Collect sinter and limestone as for skip 2

Skips 9 and 10 : Coke automatically charged
Average wait before dumping large bell at 10-ft stockline = 7 min

Total running time of car to collect one round = 10 min 50 sec

Standing time of car = 10 min

Average times per trip:	Time, sec
From skip pit to distant bin in south direction	20
Filling time	20
Returning past skip pit and to nearer bin in north direction	30
Filling time	20
Returning to skip pit	15
Dumping load	5
Pause to examine stockline recorder	5
Total	2 min

Note—After a charging delay while the scale car was greased the furnace was filled at the rate of 15 ft/hr; this is 150% of the normal charging speed required when this furnace makes 650–700 tons/day and called for no special haste by the operator)

Type B—Double-Hopper Scale Car

Furnace make (at time of test)	650 tons/day
Number of rounds per 24 hr	82
Average time per round	17 min 30 sec
Order of charge	OCOC Dump OCOC Dump
Av. running time of car to collect round	4 min 40 sec
Av. standing time of car per round	12 min 50 sec
Av. running time of compressor per round	1 min 30 sec
Centre of skip pit to south end of scale-car track	140 ft
Centre of skip pit to north end	150 ft
Max. speed of car between end of track and skip pit	670 ft/min

Av. acceleration while obtaining this speed	1·2 ft/sec/sec
H.p. to obtain acceleration	52
Motors fitted	Two 45-h.p.
Max. speed during normal operation	375 ft/min
Max. acceleration during normal operation	0·75 ft/sec/sec

Note—This furnace can make, and has made, 800 tons/day and the running and waiting times would then be approximately 5 min 20 sec and 10 min 20 sec respectively

The above performance data for single-hopper and double-hopper scale cars have been compiled from actual observations made on plants in normal operation in Britain. They show that the designs are generous as, although the single-hopper scale car, type *A*, might be described as the standard car for furnaces making 500 tons/day, the car has plenty of reserve when the output is 700 tons/day and can well manage more when this is obtained by a smooth and regular rate of furnace driving. Similarly the double-hopper scale car, type *B*, might form the standard for furnaces making 800 tons/day, but examination of the data will show that 1000 tons/day, produced with regular driving, would come well within its scope. There are as yet no examples in this country of the 1200-tons/day furnace and scale car, type *C*, but the same liberal rating is general in American practice. This leads to the conclusion that three standard types might be developed to meet the requirements of all modern furnaces of sizes that can now be foreseen. It would detract but little from the objects of standardization to vary, within reasonable limits, the volumetric capacity of the scale-car hoppers. This would permit the car hopper to correspond in size to the skip, for there may be no justification in advocating only three sizes of tub.

The foregoing figures may lead to the query as to why a hopper of 9–10 tons' capacity (150–165 cu.ft.) is used to carry loads which are normally in the region of 4–5 tons. The answer lies in the fact that all eventualities are met if the hopper capacity is equal to the skip tubs' capacity, for the car could then, in an emergency, be used to carry coke from one of the bins usually used for ore. The occasion might arise at times such as when coke-bin wearing plates have to be renewed. If the volumetric capacity of the hopper is then 150–160 cu.ft., it is wise to allow for a load of 9–10 tons in case the hopper is inadvertently or unwisely loaded with a full volume of ore.

Part III—HOT-METAL LADLES, *by D. R. Brown*

Hot-metal ladles once derived most of their characteristics from the more important features of the plant they served. Mobility and the ability to contain molten metal safely were the prerequisites ; thereafter size, shape, and type were ordained by outside influences such as headroom, clearances between tracks and fixed objects, distances to be traversed, and limitations of cranes or locomotives. This led to a variety of ladles : round pots, oval pots ; with lids, without lids ; widely flared, scantly flared ; deep or shallow ; wide protruding lips, small close-fitted lips ; barrel-shaped pots ; and cigar-shaped pots. In size they have ranged from 150 tons to less than 20 tons. Now, when the hot-metal ladles at a modern plant may be called upon each week to transport 20,000 tons of molten iron over considerable distances, the equipment becomes entitled to consideration on its own account. The opportunity to bestow this consideration occurs when new plants or major reconstructions are on the drawing board. It is then that wide experience, accumulated from the use of many types, can help to devise ladle equipment of basically sound design, suited to the particular purpose for which it is intended, and the result can be properly catered for in the layout and design of the plant.

USES OF VARIOUS TYPES OF HOT-METAL LADLE

It is intended to deal here only with those matters concerning ladles of interest in a modern plant. There is no pre-eminent, all-purpose ladle ; a wide variety in design exists—although this can be limited—to contend with the wide variety of task. This can be illustrated by taking as one extreme a single foundry-iron furnace with a short straight track leading to a pig machine, in constant readiness to deal with the product, and as the other extreme there is the extensive combined iron and steel works comprising several groups of blast-furnaces, more than one melting shop, and a separate installation of pig machines. The first illustration can well be served by simple open ladles, for heat conservation is unnecessary ; initial cost can thus be minimized and relining simplified. In the case of the other extreme, owing to the complexity of the organization, ladles may be required to hold metal for considerable periods and as the sensible heat is of value in the melting shop, heat conservation is of major importance. A still more extreme case can be visualized where the steelplants are separated from the furnaces by a distance of several miles and there may also be an absence of mixers and limited crane capacity. To cater for this last illustration there is the high-thermal-efficiency hot-metal mixer car, holding large quantities of metal, of high thermal efficiency, and capable of being easily emptied into transfer ladles of sizes suited to the melting-shop cranes. Between the plants best served by simple open-pot ladles and those best served by hot-metal mixer cars, there lies the vast majority that call for a nicely blended compromise, *i.e.*, a ladle to conserve heat without greatly adding to the initial cost or complicating the process of relining ; one that can be handled by cranes or short-poured at a pig machine, that can be filled through a reasonably sized opening, and one that offers high availability when a relined spare pot can keep all cars in constant use. In this category are several barrel-shaped ladles and the Kling ladle, which will be described later ; for brevity the whole class will be referred to as the Kling type, using the name generically.

The most suitable type having been selected, there remains the question of size. Very frequently this subject receives too meticulous care, and the fact that the internal volume of any ladle lined with refractories will not consistently remain the same is overlooked. A medium-sized ladle, when the lining is worn, can easily hold 20% more metal than when the lining is new; yet, on the other hand, the same ladle may at some time contain a skull and hold even less than when the lining is new. If the actual capacity of the ladle is somewhat ambiguous, it suggests that calculations to determine ladle size should be made more from experience than by precise mathematics. It also suggests that it is pointless to set up a too rigid specification in this respect.

If these estimates of the situation are correct it follows that developments in ladle design might be limited to three types: (1) Open-pot ladles, (2) the Kling category of ladles, and (3) hot-metal mixer cars.

It may be fair to assume that, in the period which lies immediately ahead, interest in this country will centre chiefly on furnaces in the following general categories:

(1) 400 tons/day (foundry iron from home ores)
(2) 500 tons/day (steelworks iron from home ores and foundry iron from foreign ores)
(3) 750 tons/day (steelworks iron from foreign ores)
(4) 1000 tons/day (steelworks iron from foreign ores).

If the operating programme is a minimum of five casts per day, the average casts at the above furnaces would be 80 tons, 100 tons, 150 tons, and 200 tons. But there are multifarious reasons why an individual cast may much exceed the average, so it is advisable to cater for an occasional cast 25% above normal, say, 100 tons, 125 tons, 200 tons, and 250 tons. These casts might well be carried thus:

100 tons: two 50-ton ladles
125 tons: two 75-ton ladles or one 125-ton hot-metal mixer car
200 tons: three 75-ton ladles or two 125-ton hot-metal mixer cars
250 tons: four 75-ton ladles or two 125-ton hot-metal mixer cars

If a midway example is examined it will be seen that from a 750-ton furnace, even when the output is steady, there may be casts varying between 150 and 200 tons; equally the nominal 75-ton ladle may be worn and hold nearer 90 tons of metal. Thus, as the furnace output is unlikely to be steady, there are three imponderables which combine to ridicule any mathematical approach. The way is then open for the industry to accept a limited number of sizes and to profit by a measure of standardization.

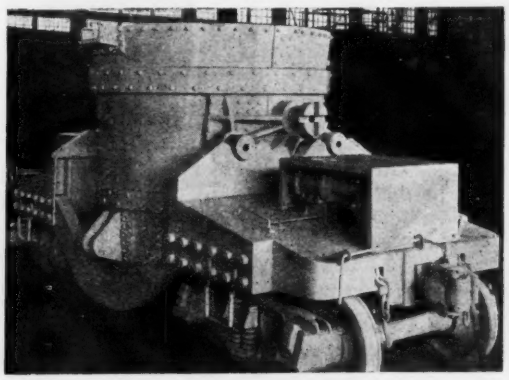


Fig. 5—50-ton hot-metal ladle (open pot)

There is certain information which will help in the selection of ladle equipment that can be derived only from practical experiments. A knowledge of comparative tractive loads imposed by different types or a comparison of their heat losses is useful, but to collect such information under working conditions on an extensive scale would interfere drastically with the routine at the plant where the tests were made. The following figures are therefore submitted with some diffidence, as the figures result from a limited number of tests taken as opportunity occurred; no claim is made to scientific accuracy but the work was sufficiently thorough to give comparisons from which correct inferences can be drawn:

Tractive Effort

	Gross Weight tons	Pull to Start, tons	Pull to Keep Moving, tons	Pull to Navigate 100-ft. Radius Curves, tons
125-ton Mixer Ladle (empty) (*see* Note I)	125	3	$1\frac{1}{4}$–$1\frac{1}{2}$	...
75-ton Kling Ladle (full) (*see* Note II)	140	$1\frac{1}{2}$	$\frac{3}{4}$	1
40-ton Open-Top Ladle (full) (*see* Note III)	60	5	$2\frac{1}{2}$	3
35-ton Open-Top Ladle (full) (*see* Note IV)	54	$4\frac{1}{2}$	2	4

Note I—Modern equipment; axle journals in plain bearings; suggest good readings result from exceptionally robust equipment.

Note II—Comparatively new equipment; wheels mounted independently on 8-in. roller bearings on fixed axles; bogie trucks respond to curves.

Note III—Old equipment; axle journals in plain bearings.

Note IV—Old equipment; axle journals in plain bearings; four-wheel carriage not so responsive on curves as bogie carriage.

Heat Conservation

	Kling Ladle (75 tons)	Open-Top Ladle (40 tons)
Furnace runner temperature, °C	1315	1325
Pig-casting-machine runner temperature, °C	1285	1200
Lapse of time, hr	$4\frac{1}{2}$	$2\frac{1}{2}$
Drop in temperature, °C/hr	7	55

Note—From an American source, the temperature drop in a hot-metal mixer car has been given as an average of $5\frac{1}{2}$° C (10° F)/hr, over a 10-hr period after the initial heating of the lining.

There are certain points which arise from the relationship between hot-metal ladles and the pig-casting machine which are worthy of consideration. Either the open-pot type or the Kling type of ladle can be fitted with short-pour horns (*see* Figs. 5–7) which, engaging with short-pour stands at the pig machine, enable the ladle to be emptied while the lip describes only a very small arc. This permits the pouring pot to be placed in proximity and so little or no splashing occurs. The lip of the hot-metal

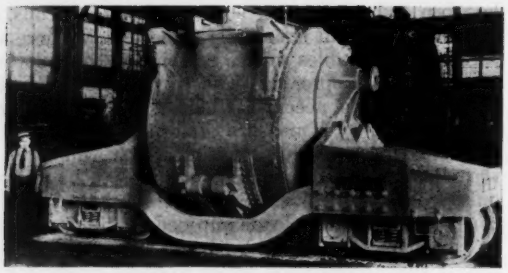


Fig. 6—70-ton hot-metal ladle (enclosed pot)

mixer car travels through a considerable arc whilst the ladle is being emptied and as the pouring pot must be located at the bottom extremity of this arc, a fair amount of splashing occurs in the early stages of pouring. The splashing is not of serious consequence, but the long pour also demands that the tail-end of the pig machines shall be at ground level, which makes access to tail-end sprockets, removal of shot, and drainage of the sump, somewhat more inconvenient than with the short-pour machine. It also necessitates a steeper inclination of the pig-machine strands which adversely affects certain principles of the design. These features weigh against the mixer car, but in its favour is the fact that no ladle-tilting device is required, for each carriage is fitted with a tilting mechanism, as will be seen from the description which follows later.

The availability factor is another point which should be taken into consideration. A number of open-pot or Kling ladles and carriages can be kept in almost constant use by having one spare ladle undergoing relining ; against this, a complete mixer car must be laid aside when relining becomes due, hence there is a bigger proportion of equipment out of service in a given time. The speed with which relining can be carried out has also some effect on the availability factor and first place must here be given to the open-pot type. Next in order is the Kling ladle, the brickwork of which requires a little more skill in laying but does not call for the use of formers. Relining a barrel-shaped ladle or a hot-metal mixer car is a slightly more complicated process and needs the use of formers or arch supports.

THE OPEN-POT LADLE (FIG. 5)

Carriage

Although small ladles can be carried on four wheels, ladles of 50 tons and upwards are carried on cars fitted with two articulated 4-wheel bogie trucks. Axle journals are mounted in plain or roller bearings. If many curves are to be negotiated, wheels mounted in roller bearings on stationary axles can be used to gain the advantage of a differential action. Arch bar frames can be of cast steel or of welded construction and the bolster is supported in the arch bar frame on a nest of springs at either side. End frames are usually of welded construction and can be made suitable for any standard type of drawbar and coupling. Side frames are made from heavy rolled steel slabs, as also are the trunnion supports upon which the pot rests. Hot-metal ladle carriages are used (i) without brakes, (ii) with brakes intended only for parking, or (iii) with brakes that can be applied whilst running, in which case a standard type of Westinghouse brake is usually employed. Where any inclines have to be negotiated it is advisable to supplement the couplings with safety chains because an accident might have extremely serious results because of the nature of the load. Another small precaution is to tile the flat upper surface of the end

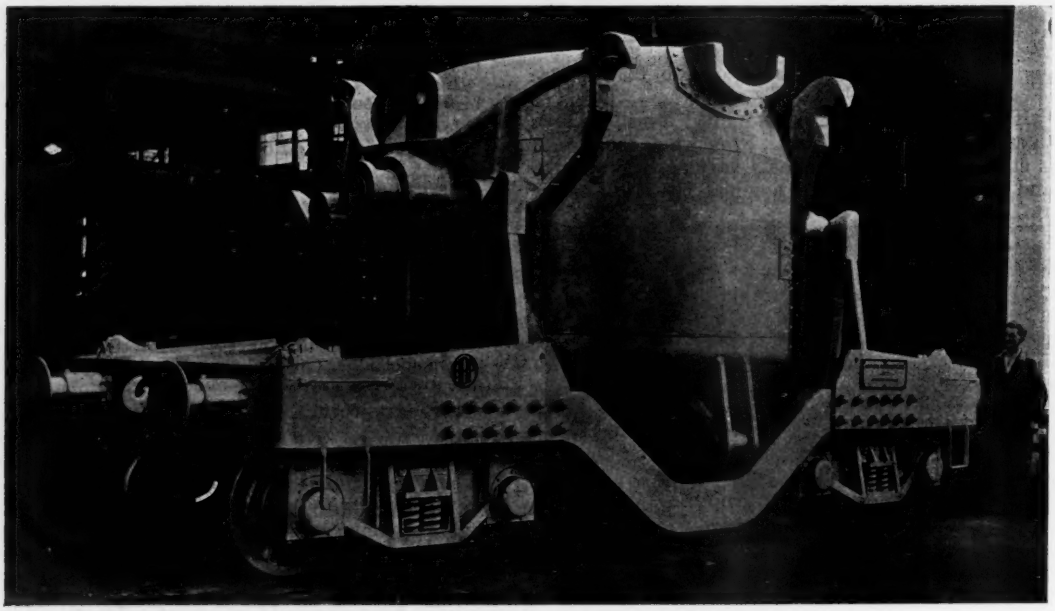


Fig. 7—75-ton Kling-type hot-metal ladle]

carriages with firebricks to avoid serious damage if a ladle is moved from below a runner spout whilst hot metal is still dropping.

Ladle Pot

The pot consists of a cylindrical vessel somewhat larger in diameter at the top than at the bottom, built of either riveted or welded mild-steel plate, the bottom being sometimes dished at the edges. The trunnion assemblies consist of heavy steel castings, each having a pair of short-bearing trunnions upon which the ladle rests whilst in the carriage; the crane-hook trunnions are placed above the centre of gravity and protrude beyond the plane of the bearing trunnions. For short-pouring at a pig machine, the open-pot ladle can be fitted with horns combined with the trunnion castings similar to those shown in Fig. 7. The lip or spout of the modern ladle is a small fitting, lined with refractories, which can be readily renewed. Ladle pots are fitted with tilting lugs to take the auxiliary hook when the ladle is being tilted by crane. Open-ladle pots can be fitted with lids for retaining heat, but the result is not so satisfactory as pots designed for heat conservation.

KLING-TYPE LADLE (FIGS. 6 AND 7)

The following description is intended to embrace all types of heat-conserving ladles which are lifted from the carriage and handled by overhead cranes.

Carriage

The carriage used with Kling-type ladles is in all respects identical to that used with open-pot ladles and described in the preceding section.

Ladle Pot

A typical example of the barrel-shaped ladle is shown in Fig. 6. The pot has a capacity of 70 tons and is built as a mild-steel cylinder having dished ends. This pot cannot be inverted with a fixed tilting lug but the same effect is achieved by attaching the crane hook to a gall chain which is anchored at the opposite side of the ladle; the device can be used from either side and is clearly seen in Fig. 6. Short-pour horns, when fitted, are attached to the ladle body and do not form extensions of the trunnion castings.

The Kling ladle proper is a cylindrical pot with an almost hemispherical dished bottom. The top section of the pot has a slight inward slope on the axis across the spouts; on the other axis the inward slope is much more pronounced and results in an elongated, oval opening with a short spout at either end. The trunnion castings are similar to those used with open-top pots of a similar capacity. Fixed tilting lugs are fitted on either side. The most modern pots are made in welded construction, the whole being stress-relieved as a safety precaution.

Fig. 8—Hot-metal mixer car

HOT-METAL MIXER CARS (FIG. 8)

A hot-metal mixer car does not consist of a distinct carriage and pot, as the mixer-ladle car is built without an underframe. The ladle is equipped at either end with long trunnions, each having two journals. These journals rest in bearings on the end frames which are otherwise entirely independent of each other. According to the capacity of the ladle, the end frames are carried by 4, 6, 8, 10, or 12 wheels in either single or compound bogie trucks. When more than six wheels are used the end frames rest on intermediary bolsters which in turn rest on the two bogie trucks. Wheels, axles, and trucks are of conventional design and either plain or roller bearings can be used.

The cigar-shaped ladle with its elongated trunnions at either end can, owing to the absence of an underframe, be rotated through 360° by a dumping device consisting of an electric motor directly connected to a worm-gear reducing unit and then through sets of gears to the trunnion. Although the worm gear is self-locking, the axis of the trunnions is above the true axis of the ladle so that under all conditions the ladle will tend to remain upright or will right itself if the gear should break during pouring. The dumping device can be hand-operated in the event of a current failure.

Hot-metal mixer cars are built in various sizes from 60 to 200 tons. Heat conservation is excellent, especially with the larger sizes. This characteristic, which is common to all the enclosed types of ladles, has an effect on the life of the lining which is not exposed to so drastic changes in temperature as is the lining of an open pot.

Part IV—SLAG LADLES, *by T. H. Stayman*

THERE are two main types of ladles for handling slag: (1) Ladles tipped by the locomotive through a chain and pulleys, and (2) steam- or air-tipped ladles. The first of these is well known and needs no description in this paper; it is therefore proposed to deal only with the steam- or air-tipped type.

GENERAL DESCRIPTION

Figure 9 shows a general arrangement of this type of ladle and carriage.

The ladle is carried in a cast-steel trunnion ring at each end of which is keyed a vertical toothed wheel arranged to mesh with a horizontal rack, the load being carried by vertical rollers running on a pedestal mounted on the main frame. A steam- or air-operated cylinder is mounted horizontally on the main frame at one end of the car and guided by two rods placed one above and one below the cylinder. Provision is made at each end of the cylinder for a

U-shaped key which engages a loose collar on a wrist pin attached to the vertical toothed wheel.

Steam or air is admitted from the main line through a 1¼-in. control valve to the hollow piston rod and, on operation of the control-valve lever, moves the cylinder in a predetermined direction. This operates the vertical toothed wheel, tips the ladle, and at the same time moves it across the car.

MATERIAL

Slag ladles may be of cast steel, or hematite, or a special iron to the following analysis :

Si	2·1–2·5%	Mn	0·5–0·9%
S	0·10(max)%	T.C.	2·9–3·4%
P	0·20(max)%	Cr	0·3–0·7%

Cast-steel ladles have given good results but it has been reported that firms using steel ladles have more trouble with sticking than others using iron ladles. Improved results have been obtained by grinding the inside of cast-steel ladles before use.

Hematite ladles are commonly used in the chain-tipped arrangement and have given good results over a number of years, but owing to the low tensile strength of hematite, they are cast thicker and heavier than cast steel, thus putting extra work on the tipping gear and additional load on the car. The weight of hematite ladles is approximately twice that of steel ladles.

The special iron mentioned above gives a tensile strength of 35,000–45,000 lb/sq.in., and a ladle has been designed with a weight only 5% above that of cast steel.

Of sixteen ladles at present under construction for one firm, eight are in steel and eight in special iron. All the ladles are to work under the same conditions and the results will be carefully noted.

FEATURES OF CONSTRUCTION AND OPERATION

Ladle

The ladle is attached to the trunnion ring by eye-bolts or pins. These are loose connections and are so designed to allow for expansion of the ladle due to heating and cooling. With rigid connections between ladle and ring, changing heat conditions cause undue strain on the ring.

Tipping Cylinder

The tipping cylinder is of larger diameter than necessary for normal working conditions so that when conditions at the tip are unfavourable, *i.e.*, low pressure for operation or the ladle only half full of slag, ample power is available for tipping.

The cylinders are designed to work at 100 lb/sq.in., but they have operated successfully on pressures as low as 40 lb/sq.in.

Steam or Air Requirements

The first step is to determine the cubic capacity of the tipping cylinder. To allow for rapping ladles, loss of air or steam in pipe lines, and loss due to cold weather, the air or steam supply should be three times the total cylinder capacity of all the cars in the train. Diesel locomotives seldom have sufficient air for a number of slag ladles, nor do they have room for mounting additional air reservoirs. Most plants having steam-operated cars depend on the locomotive for the steam and in this case it is frequently necessary to halt operations to build up steam pressure.

Piston Types

Cars for steam operation exclusively have usually been fitted with pistons having three piston rings, while cars for air operation have cup leathers attached

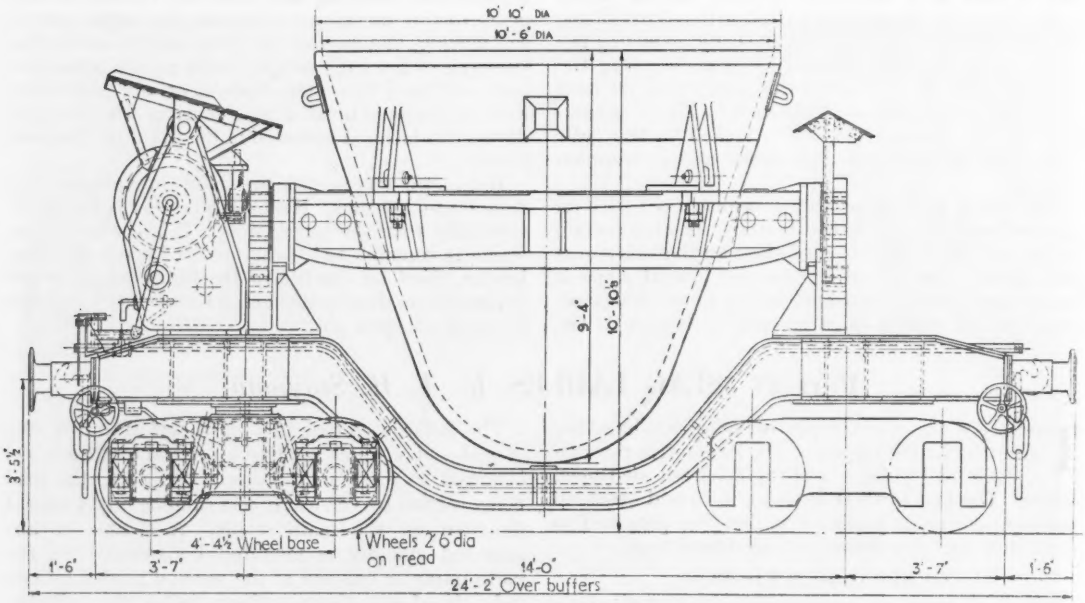


Fig. 9—General arrangement of slag ladle tipped by air or steam cylinders

to the piston. A piston which is suitable for operation with either air or steam has been designed and is in use.

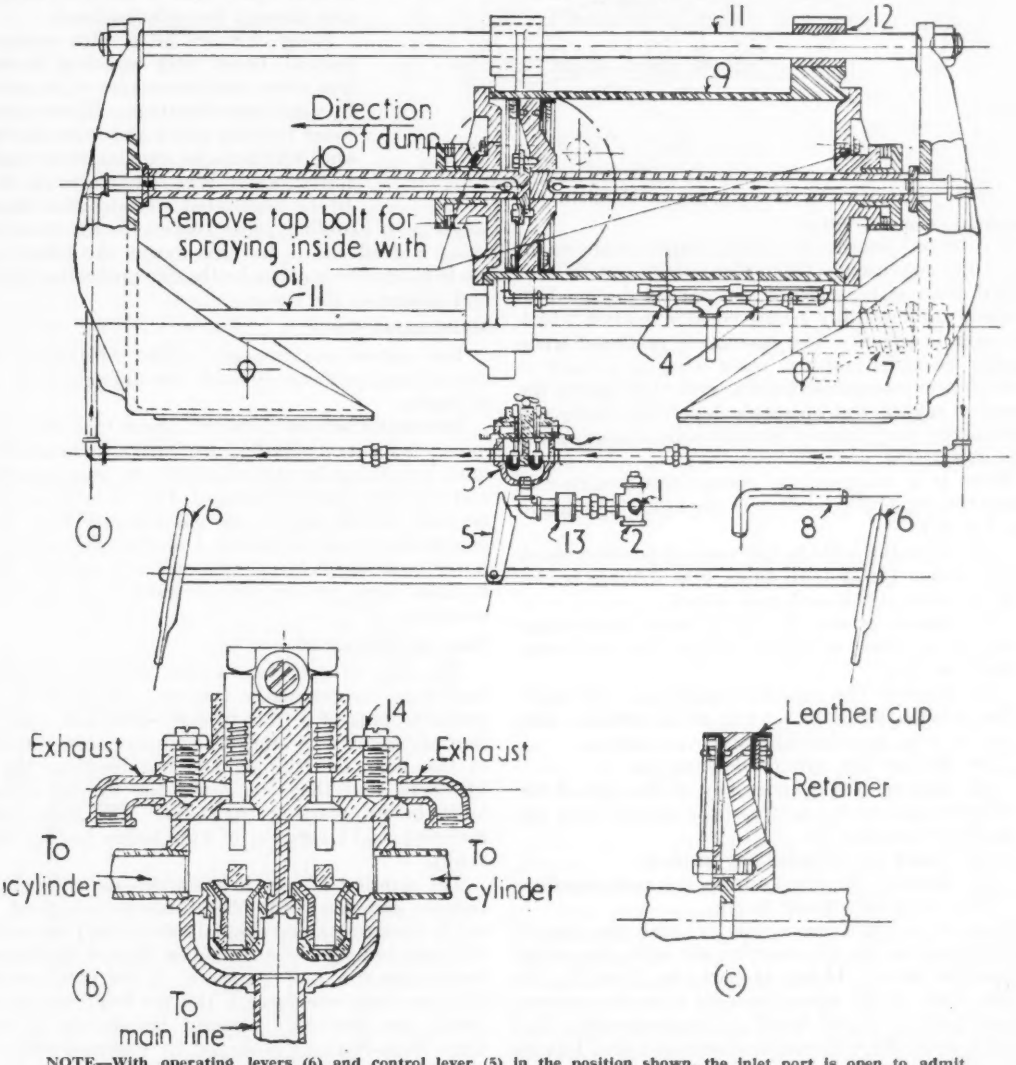
Steam *versus* Air Power

Where the slag cars are hauled by steam locomotives it will usually be found more convenient to tip the ladles by steam as the locomotive is not likely to have sufficient air to tip the ladles at all times. In addition there is less danger of freezing with steam than with air. When Diesel locomotives are used or where the locomotive cannot supply the steam required, air-operated cars become necessary. Air can be supplied from a compressor car consisting of a flat car on which are mounted two air compressors and two air receivers plus shelter for the crew. This car may remain at the dump or it may be hauled back and forth in the slag train.

Speed of Tipping

The speed of tipping was originally controlled by means of an oil cataract cylinder which was fastened to, and moved with, the tipping cylinder. Control was obtained by means of a needle valve which regulated the amount of oil passing from one end of the control cylinder to the other.

Because the operators failed to use this valve a diaphragm with a small hole has been fitted between two flanges in the oil by-pass pipe. This is theoretically satisfactory but again it depends on the operator keeping the cylinder full of oil.



NOTE—With operating levers (6) and control lever (5) in the position shown, the inlet port is open to admit air to the dumping side of the cylinder. The locking pin (8) is shown withdrawn from the hole in the pedestal

Fig. 10—Automatic control valve and cylinder: (*a*) For air operation (*b*) Enlarged view of control valve (3), inlet and exhaust valves are shown in the neutral position which they take when levers (5) and (6) are vertical and plug cock (2) is closed (*c*) Enlarged view of piston

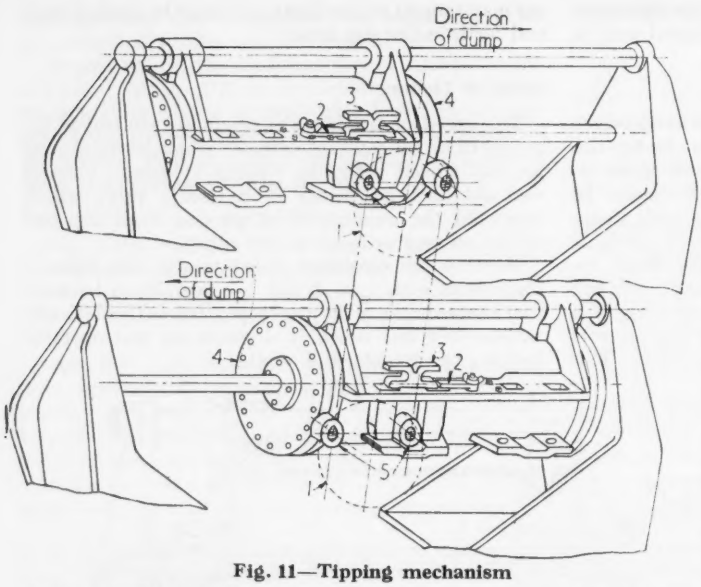


Fig. 11—Tipping mechanism

Automatic Control Valve

A patented automatic control valve as shown in Fig. 10 is now used. This valve, which can be controlled from either side of the car, is designed to produce automatically an air cushion against which the ladle is tipped. The cushion is produced when tipping the ladle and also when bringing it back to vertical, the arrangement being such that should the operator release the operating lever, the ladle will come to rest in a balanced position without completing its travel.

When it is necessary to change the direction of dump this can easily be done in the following manner (*see* Fig. 11) :

(1) With the ladle in the vertical position chock rack wheel (1) at both sides, and if there is slag in the ladle chock each rack wheel.

(2) Loosen clamp (2) which holds the cylinder key (3) in place, swing the clamp clear, and withdraw the key.

(3) Remove the cylinder-locking pin and move the cylinder to the other end of its stroke. This can be done by admitting air as in tipping.

(4) Replace the cylinder-locking pin.

(5) Replace the cylinder key at the end of the cylinder nearest the centre of the car and over the wrist-pin bushing (5).

(6) Clamp the cylinder key in place.

(7) Remove the chocks from the rack wheels.

The car is then ready to tip.

There is no rigid connection between the tipping mechanism and the ring carrying the ladle, the simple connection being a **U**-key as shown by (3) in Fig. 11.

The angle of tip when the ladle is in the extreme tipped position should be 45° to the horizontal. This angle is generally sufficient to ensure the skull leaving the ladle and results in less rapping of the ladle against the underframe than with ladles having a smaller angle of tip. This minimizes wear on the car and reduces the time the car is at the tip. While the steep angle of tip reduces the number of stickers to a minimum it is necessary at times to rap the ladle, and component parts are designed with this service in mind. To obviate further injurious shocks from rapping, heavy springs are provided at each side of the tipping cylinder. As these shocks are transmitted to the brasses in the journal box the springs appreciably increase the life of the brasses.

In steam-operated cars the tipping cylinder is lubricated through a grease cup on top of the cylinder, but as air-operated cars have pistons with cup leathers, provision is made for lubrication through the cylinder head.

Many cars are fitted with cast-steel journal boxes with standard brasses, but there are advantages to be gained by using roller bearings. These include lower tractive effort and less maintenance which can be set against the higher initial cost. When sharp curves have to be negotiated consideration should be given to installing roller bearings in the wheel and using a dead axle. By this means the wheels are made to revolve independently, thus reducing friction and sliding on the track.

Maintenance Costs

The cost of maintenance varies with local conditions and particularly with the differences in size of plants.

To a manufacturer it would appear that very little money is spent on maintenance, except for lubrication, until something breaks, when the car goes for repair and is given a general overhaul. Cars should, however, be sent periodically to the maintenance shop for a general check and overhaul. Unfortunately operators never seem to have enough cars and cannot afford to take them out of service until it is absolutely necessary.

Time in Operation

The time in operation varies with the length of haul from the furnace to the tip. The time for the actual tipping of a train of eight 400-cu.ft. cars was recently taken as 8 min. The tipping crew consisted of two men, one tipping and one bleeding the air cylinders, and they completed the tipping of each ladle before starting the next. The time taken included slight rapping of two ladles having small skulls.

The number of cars in operation depends on the number and size of the blast-furnaces in a plant. A single furnace making about 1200 tons of iron per day will require nine 400-cu.ft. ladles, four at the furnace, four in the run, and one spare. If the plant consists of more than one furnace the car requirements decrease per furnace. A recent installation of three large blast-furnaces is served by eighteen 400-cu.ft. slag ladles, *i.e.*, six per furnace, as compared with nine for the single furnace mentioned above. The actual needs can perhaps best be summarized as four 400-cu.ft. ladles at the furnace all the time.

Part V—INGOT BOGIES, *by T. H. Stayman*

INGOT bogies are used for the transport of bottom plates and ingot moulds, the moulds being filled while on the bogies. The bottom plates are located endwise by solid stops at the end of the wagon. Holes are often provided in the side skirting for the insertion of crane hooks, etc., to assist propulsion.

The bottom plates, which may be up to 10 in thick, the ingot moulds, and the ingots themselves, frequently have a total weight of 90 tons or more. The carrying of such a load at high temperatures presents numerous problems.

CONSTRUCTION OF INGOT-BOGIE BODIES

The ingot-bogie bodies may be of cast, riveted, or welded construction. Cast steel has been most generally used in the past, but in recent years welded bodies have given satisfactory service. It is commonly found that a cast-steel body is liable to crack, the cracks occurring across the top plate or, more frequently, in the side skirting. In the author's experience, welded bodies have not given the same trouble. It is probable that residual internal stresses in the casting, even after annealing, are more severe than those in the welded body, as, owing to the thickness of the plates in the body, distortion due to welding is not unduly severe. This distortion is, of course, further lessened by stress-relieving of the completely fabricated body.

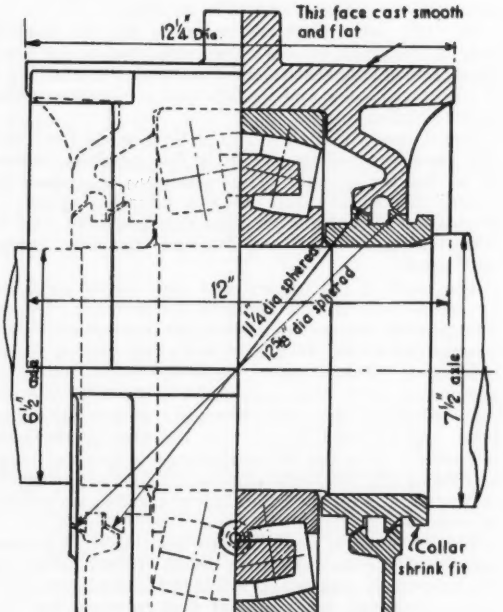


Fig. 12—Typical roller-bearing axle box, showing labyrinth rings and throwers fitted to both sides of the box

Particulars of a typical cast-steel bogie are as follows:

Wheelbase	6 ft 0 in
Height from rail to top ...	3 ft 2 in under load
Overall length	12 ft 4⅝ in
Length between bottom-plate stops	10 ft 10 in
Overall breadth	6 ft 0 in
Deck	2 in thick
Side and end skirting ...	1¼ in thick
Crossbars (six)	1⅜ in thick
Longitudinal webs (two) ...	1½ in thick
Longitudinal flanges	4½ x 2 in
Fillet size* (generally) ...	1½-in radius

* Particular attention is drawn to the size of the fillet as small fillets tend to cause cracking.

For a welded bogie of similar overall size the corresponding dimensions are as follows:

Deck	2 in thick
Side and end skirting	1¼ in thick
Crossbars (six)	1½ in thick
Longitudinal webs (two)	1¼ in thick
Longitudinal flanges	4½ × 2 in

A triangular fillet 6 × 6 × 1½ in was welded into each of the side and end skirting corners.

Particulars of the welds are as follows:

End skirting to side skirting	¾-in fillet inside, ¾-in single V butt outside
Deck plate to skirting and longitudinal	¾-in fillet inside and outside
End skirting to longitudinals	¾-in fillet both sides
Deck plates to crossbars	½-in fillet both sides
Side skirting to crossbars	½-in fillet both sides
Longitudinal flange to longitudinal web	¾-in fillet both sides
Longitudinals to crossbars	½-in fillets (four)

Riveted bogies are of similar proportions to welded bogies, members being connected by heavy angles with correspondingly strong riveting.

WHEEL BEARINGS

A major problem is, of course, the provision of satisfactory wheel bearings. Axle boxes are internal, *i.e.*, they are fitted between the wheels. A typical roller-bearing axle box is depicted in Fig. 12. Labyrinth rings and throwers are fitted to both sides of the box. The axle-box casting is usually split to permit the lowering of the bottom half for inspection purposes. A wooden roller held against the revolving journal ensures lubrication of the white-metal bearing which is cast integral with the top block.

As alternatives, ball or roller bearings may be fitted inside the wheel itself. Such an arrangement with roller bearings is shown in Fig. 13. The back plate and cover are bolted on to the wheel, the labyrinth ring being pressed against a shoulder on the axle. The axle is clamped to the body and does not rotate. This arrangement permits wheels on the same axle to rotate at different speeds, and may be considered advantageous when negotiating sharp curves.

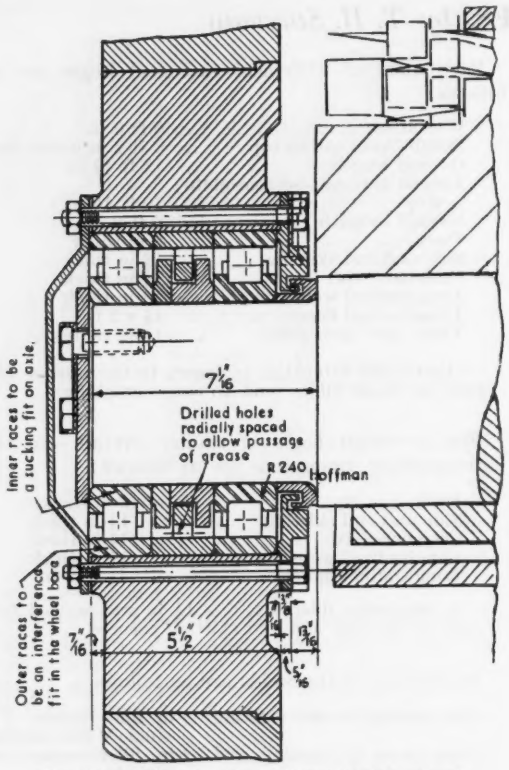

Inner races to be a sucking fit on axle.
7 1/16"
Drilled holes radially spaced to allow passage of grease
R 240
Hoffman
Outer races to be an interference fit in the wheel bore
7 1/16"
5 1/2"

Fig. 13—Roller bearings fitted inside wheels

In all cases the bearing manufacturers should be consulted with regard to lubricants used, because of the high temperatures involved. Oil lubrication may be effected by means of pipes fed from the end of the bogie, or directly into the axle box. Suitably located nipples are used in the case of grease.

Shock loading due to the slipping of moulds, etc., should always be considered when determining bearing sizes, although such shocks are lessened by the bearing springs. These are usually of the triple-helical type, and for the cast-steel car particulars are as follows :

Outside Coil : 2¼-in square section, 11⅞-in outside dia, 4½ coils, R.H.
Middle Coil : 1⅜-in square section, 7½-in outside dia, 7¾ coils, L.H.
Inner Coil : ¾-in square section, 4-in outside dia, 13 coils, R.H.

DRAW GEAR

Draw gear is of the continuous type, there generally being one helical draw-spring in a central cradle carried on a light fabricated support. Forged steel tension bolts transmit the pull to the cradle either direct or through cast-steel buckles which carry the drawbar pins. Bogies are connected by cast or forged links into which the drawbar pins fit. As it is necessary to shield the buckles and pins from spillage of metal, they are usually fitted under a protruding block on the body, which also acts as a buffer. It should be possible to couple and uncouple bogies easily without having to stand between them. This can be done by means of a pivoted hand lever, one end of which carries the drawbar pin. The pin is held in the engaged position by the greater weight of the outer part of the lever. In the other position it is held by a spring clip on the end of the body.

Acknowledgments

The authors gratefully acknowledge the courtesy and assistance of the managements at those plants where operational data were collected, and particularly those of Messrs. Ashmore, Benson, Pease and Co., Ltd., and Messrs. Head, Wrightson and Co., Ltd., who provided the majority of the technical information.

Seventy-Five Years Ago

An important aspect of the Special Summer Meeting of the Institute at Liége, in August, 1873—the Institute's first foreign meeting—were the visits to local works. On Wednesday, 20th August, 1873, the members were taken by special steamboats to the works of the Société John Cockerill and Co., at Seraing. In a pamphlet presented to each member, the Seraing works were described as the largest individual works in the world, with the exception of the Essen Works. There were concerns in England employing a larger number of hands, but their works were widely scattered, while the Cockerill company was, so to speak, under one roof.

The foundation of the Seraing establishments was due to John Cockerill, who was born at Haslingden, in Lancashire, in 1790. His father had introduced at Liége the construction of machines for wool spinning, and having acquired there a great fortune, he left his factories to his two sons, James and John, in 1813.

In 1822 John Cockerill came to live at Seraing, bought the grant of coal mines upon which the works stood, and introduced to the Continent the smelting of cast iron with coke, and the fabrication of iron according to the English process.

The Cockerill Company, at the time of the Institute's visit, was managed by a board of five members, assisted by a director-general. The operations comprised the mining of coal and iron ore, the reduction of ore, the fabrication of cast and wrought iron and steel, and the construction of machines, engines, and iron bridges and vessels.

The work of the factory was then chiefly supported by orders from the Netherlands Government. In 1824 "magnificent steam engines of 300 horse-power" were constructed under John Cockerill's supervision for the men-of-war of the country, while the English navy had engines of only 150 horse-power. In 1826 the first coke blast-furnace, the furnaces, rollers, hammers, blast engines, and engines for the iron factory were introduced. The first Belgian locomotive was constructed in 1834 by the Seraing Works, which soon after supplied all the engines for the beginning of the Belgian network of railways.

The fortunes of the company continued to increase rapidly until John Cockerill's death, in 1840. The joint-stock company established in 1842 to continue the operations comprised the grant of coal mining with three collieries, thirty-seven coke kilns, two blast-furnaces, a vast iron-smelting and a copper foundry, and an iron factory with five reverberatory furnaces, five sets of rolls, hammers, steam engines, etc., and in addition a great many forge furnaces and machine tools. There were 2200 workmen.

N E W S

ANNOUNCEMENTS AND NEWS OF SCIENCE AND INDUSTRY

THE IRON AND STEEL INSTITUTE

Autumn Meeting, 1948

The Autumn Meeting of The Iron and Steel Institute will be held on Wednesday and Thursday, 10th and 11th November, 1948, under the Chairmanship of the President, Sir Andrew McCance, D.Sc., F.R.S. The Sessions will take place at the Offices of the Institute, 4 Grosvenor Gardens, London, S.W.1.

Buffet Luncheons (price 6*s.* each per person) will be arranged in the Library of the Institute between the Morning and Afternoon Sessions on both days. Buffet Luncheon tickets will not be issued in advance, but Members will be asked to buy them on entering the Library.

The Council intends to give formal notice at this Meeting of a proposal to amend certain By-Laws at the Annual General Meeting in 1949.

The programme of the Meeting is as follows :

Wednesday, 10th November, 1948

10.0 a.m. to 1.15 p.m.—Morning Session

10.0 A.M. to 10.30 A.M.—Formal business including notice of a proposal to amend By-Laws Nos. 3 and 21.

10.30 A.M. to 11.30 A.M.—Discussion on :
"*A Note on the Varying-Turbulence Cowper Stove.—The Denain-Anzin Tests and the C.S.I. Standard Cowper Stove*," by D. Petit (*October*, 1948).

11.30 A.M. to 12.15 P.M.—Discussion on :
"*Magnetic-Concentration Experiments upon Iron Ores in North Lincolnshire Practice*," by L. Reeve (*July*, 1948).

12.15 P.M. to 1.15 P.M.—Discussion on :
"*The Work and Organization of a Statistical Department in Heavy Industry, with Particular Reference to the Steel Industry*," by A. W. Swan (*September*, 1948).

1.15 P.M. to 2.30 P.M.—Buffet Luncheon in the Library of the Institute.

2.30 p.m. to 5.0 p.m.—Afternoon Session

2.30 P.M. to 3.30 P.M.—Discussion on :
"*The Application of Oxygen-Enriched Air to Side-Blown Converter Practice*," by J. L. Harrison, W. C. Newell, and A. Hartley (*July*, 1948).

3.30 P.M. to 4.15 P.M.—Discussion on :
"*Development of an Improved Basic Bessemer Steel*," by H. A. Dickie (*August*, 1948).

4.15 P.M. to 5.0 P.M.—Discussion on :
"*Ingot Structure in a Series from Rimmed to Killed Steel made from the Same Cast*," by P. M. Macnair (*October*, 1948).

Thursday, 11th November

10.0 a.m. to 1.15 p.m.—Morning Session

10.0 A.M. to 10.45 A.M.—Discussion on :
"*Ingot Surface Defects in Structural Steels*," by L. Reeve (*October*, 1948).

10.45 A.M. to 11.45 A.M.—Joint discussion on :
"*Open-Hearth Furnace Instrumentation*," by E. Rogers (*September*, 1948).
"*First Report of the Open-Hearth Instruments Sub-Committee*," by F. L. Robertson (*September*, 1948).

11.45 A.M. to 12.30 P.M.—Discussion on :
"*Rapid Estimation of Slag Basicity*," by W. A. Smith, J. Monaghan and W. Hay (*October*, 1948).

12.30 P.M. to 1.15 P.M.—Discussion on :
"*The Determination of Moisture in Producer Gas*," by J. Pearson and R. Toye (*August*, 1948)

1.15 P.M. to 2.30 P.M.—Buffet Luncheon in the Library of the Institute.

2.30 p.m. to 5.0 p.m.—Afternoon Session

2.30 P.M. to 3.30 P.M.—Discussion on :
"*Fatigue Tests on Crankshaft Steels*," by P. H. Frith, (*August*, 1948).

3.30 P.M. to 4.15 P.M.—Discussion on :
"*A Magnetic Study of the Structure of Cold-Worked Iron–Nickel–Chromium Alloys (Stainless-Steel Wires)*," by P. T. Hobson, E. S. Chatt, and W. P. Osmond (*June*, 1948).

4.15 P.M. to 5.0 P.M.—Discussion on :
"*The Interrelation of Hardenability and Isothermal Transformation Data*," by W. I. Pumphrey and F. W. Jones (*June*, 1948).

Note—The issues of the *Journal* in which the papers appear are given in brackets.

National Certificates in Metallurgy

REPORT BY THE JOINT COMMITTEE ON THE PROGRESS OF THE SCHEME FOR THE AWARD OF NATIONAL CERTIFICATES IN METALLURGY FOR THE YEAR 1947–1948

During the year under review the Institution of Metallurgists has joined The Iron and Steel Institute, the Institution of Mining and Metallurgy, and the Institute of Metals, in co-operating with the Ministry of Education in operating the scheme for National Certificates in Metallurgy in England and Wales.

Schemes for courses leading to the Ordinary and Higher National Certificates in Metallurgy have been approved by the Joint Committee and have been in operation during 1947–1948 at the following Technical Colleges :

Ordinary Certificate — Battersea Polytechnic, London ; Birmingham Central Technical College ; Chesterfield Technical College ; Coventry Technical College ; Cumberland Technical College, Workington ;

Derby Technical College; Dudley and Staffordshire Technical College; Enfield Technical College; Ilkeston Technical Evening Institute; Merchant Venturers' Technical College, Bristol; Newport Technical College; Rotherham College of Technology; Scunthorpe Technical College; Smethwick, The Chance Technical College; Swansea Technical College; Wednesbury County Technical College; and Wolverhampton and Staffordshire Technical College.

Higher Certificate—Battersea Polytechnic, London; Birmingham Central Technical College; Chesterfield Technical College; Middlesbrough, Constantine Technical College; Rugby College of Technology; Rutherford College of Technology; Wednesbury County Technical College; and Wolverhampton and Staffordshire Technical College.

Final Examinations have been held in 1948 at the following Technical Colleges for Students who satisfied the conditions laid down in Ministry of Education Rules 111, under which the scheme is operated:

Ordinary Certificate — Battersea Polytechnic, London; Birmingham Central Technical College; Chesterfield Technical College; Coventry Technical College; Cumberland Technical College, Workington; Derby Technical College; Enfield Technical College; Ilkeston Technical Evening Institute; Newport Technical College; Rotherham College of Technology; Scunthorpe Technical College; Smethwick, The Chance Technical College; Swansea Technical College; Wednesbury County Technical College; and Wolverhampton and Staffordshire Technical College.

Higher Certificate—Battersea Polytechnic, London; Birmingham Central Technical College; Chesterfield Technical College; Middlesbrough, Constantine Technical College; Rugby College of Technology; Rutherford College of Technology; Wednesbury County Technical College; and Wolverhampton and Staffordshire Technical College.

Of the 123 candidates that entered, 74 have qualified for the award of an Ordinary National Certificate in Metallurgy; 38 candidates entered for the award of a Higher National Certificate in Metallurgy, and 22 have qualified.

Distinctions have been awarded to 13 candidates who have shown an exceptional grasp of their subjects, indicating a high degree of training and knowledge in the particular subject in which the Distinction has been gained.

Prizes, taken in books and distributed by the Technical Colleges concerned, were awarded to 19 successful candidates in the final examinations in 1947, from the Prize Fund established for this purpose by The Iron and Steel Institute, the Institution of Mining and Metallurgy, and the Institute of Metals. Prizes will similarly be awarded to 17 successful candidates who have shown particular merit in the Final Examinations held in 1948.

The Joint Committee is pleased to note the increase in the number of candidates who entered for the Final Examinations for the Ordinary and Higher National Certificates in Metallurgy this year, and that there is a still greater increase in the number of students entering the first and second years of the courses.

NEWS OF MEMBERS

▶ Mr. H. M. ASPINALL has left the United Steel Companies, Ltd., Stocksbridge, and is now a metallurgist in the Department of Development and Research of the Dominion Steel and Coal Corporation, Ltd., Sydney, Nova Scotia.

▶ Dr. G. E. A. BRAMLEY has been appointed chief metallurgist at the Aluminium Wire and Cable Co., Ltd., Port Tennant Works, Swansea.

▶ Mr. B. CHALMERS has left the Atomic Energy Research Establishment to become professor of physical metallurgy at the University of Toronto.

▶ Mr. F. JAMES CLARK has been appointed general works manager of Newman Industries, Ltd., of Yate, near Bristol.

▶ Mr. A. CROSBY, previously with the Control Commission for Germany, is now a metallurgist at Frigidaire, Ltd., London.

▶ Mr. FRANK W. DALE has been appointed divisional chemist with the British Electricity Authority, Merseyside and North Wales Division.

▶ Mr. W. HOWELL DAVIES is now lecturer in metallurgy and chemistry at Luton Technical College.

▶ Mr. H. E. DIXON has left the British Iron and Steel Research Association to join the British Welding Research Association.

▶ MR. M. K. HALDAR has left the National Pipes and Tubes Co., Ltd., Shamnager, West Bengal, and has joined the Textile Machinery Corporation, Ltd., as research officer to organize the research department of that Corporation.

▶ Mr. W. H. B. HATTON has been elected to the Board of Glynwed Tubes.

▶ Mr. RUSSELL JAY has resigned his position as research metallurgist at the Mond Nickel Co., Ltd., and has joined the staff of the Morgan Refractory Co., Ltd., Battersea.

▶ Mr. H. D. JONES has been awarded the degree of B.Sc. (Hons.) in metallurgy at King's College, Newcastle-upon-Tyne, and has joined the Central Research Department of Messrs. Dorman, Long and Co., Ltd., Middlesbrough.

▶ Dr. ROBERT F. MEHL has been appointed to the Advisory Committee to the Secretary of Commerce of the United States National Bureau of Standards.

▶ Mr. MAURICE MILNE has been appointed senior engineer in the chief engineer's department of the Crawley Development Corporation.

▶ Dr. F. R. MORRALL, who for the last four years has been group leader of the Metal Trades Laboratory of the Technical Service and Development Division of the American Cyanamid Co., Stamford, Conn., has been appointed associate professor of materials engineering at the College of Applied Science, Syracuse University, Syracuse, N.Y.

▶ Mr. W. P. OSMOND has left the Research Department of Messrs. Boosey and Hawkes, Ltd., and has joined the Magnetic Tapes Division of Durex Abrasives, Ltd., of Birmingham.

▶ The Rt. Hon. The Viscount RIDLEY has joined the Board of Messrs. Head, Wrightson and Co., Ltd.

▶ DR. EARL C. SMITH has been awarded the Lamme Medal of the Ohio State University.

▶ Mr. W. D. WALKER has left the Clyde Alloy Steel Co., Ltd., Motherwell, and has taken a position in the Metallurgy Department of the University of the Witwatersrand Milner Park, Johannesburg, South Africa.

Obituary

The Council regret to record the deaths of the following:

Mr. ANDREW CORMIE BERNARD, Manager of the Carron Company, on 26th August, 1948.

Lord GREENWOOD, of Messrs. Dorman, Long and Co., Ltd., on 10th September, 1948, aged 78.

Mr. C. L. KINNEY, of Messrs. Lewis B. Kinney and Associates, Denver, Colorado, on 15th May, 1948.

Mr. THOMAS OSWALD LEWIS, General Manager and Director of Messrs. Richard Thomas and Baldwins, Ltd., Swansea, on 21st August, 1948, aged 64.

CONTRIBUTORS TO THE JOURNAL

W. A. Smith, B.Sc., A.R.T.C.—Manager of the Department of Research and Technical Development (Scottish Section) of Messrs. Stewarts and Lloyds, Ltd., Coatbridge. Mr. Smith joined Messrs. Stewarts and Lloyds in 1935.

J. Monaghan, B.Sc., A.I.M.—Technical development metallurgist in the Department of Research and Technical Development (Scottish Section) of Messrs. Stewarts and Lloyds, Ltd., Coatbridge. Mr. Monaghan was born at Birkenhead in 1922 and educated at Coventry Grammar School. He received his technical education at Wednesbury and Birmingham Technical Colleges and obtained a B.Sc. in metallurgy of London University with first-class honours in 1946. Joining Messrs. Stewarts and Lloyds' Bilston Works in 1939, Mr. Monaghan took up his present position in the Research Department on graduation, and has since been employed in the development and application of methods of slag control.

W. Hay—Senior analytical chemist, Department of Research and Technical Development (Scottish Section) of Messrs. Stewarts and Lloyds, Ltd., Coatbridge. Born in 1922, Mr. Hay was educated at Airdne Academy and at Coatbridge Technical College. He joined Messrs. Stewarts and Lloyds in 1939.

Daniel Petit—Consulting engineer and founder of the Société de Technique Industrielle, in France, and of International Research and Constructions, Ltd., in England. M. Petit introduced the "free-expansion" coke oven, which has been widely used in France for the reconstruction of coke works and also a new low-temperature carbonization oven for the production of semi-coke which, when added to high-volatile coals, will make good metallurgical coke. In 1938 he presented a paper to the British Ceramic Society on the calculation of refractory structures.

P. M. Macnair, B.Sc., Ph.D., A.R.I.C., F.I.M.—Metallurgist at Messrs. Richard Thomas and Baldwins, Ltd. Dr. Macnair was educated at the Allan Glen High School of Science, and at the University and the Royal Technical College, Glasgow. From 1915 to 1919 his studies were interrupted by war service in the Royal Flying Corps and in the Royal Air Force. During the summer of 1920 he was in Spitsbergen for five months prospecting for minerals. On graduating B.Sc. in metallurgy in 1921 Dr. Macnair was awarded an 1851 Exhibition Scholarship with which he proceeded to Sheffield University to carry out researches on slag–metal reactions, for which he received the Ph.D. degree in 1924. For the two following years he was engaged as lecturer in the Metallurgy Department at Swansea University, and in 1926 he joined the Grovesend Steel and Tinplate Company as metallurgist. Since that time, except for three years on furnace development with Messrs. J. Howden and Co., in London, he has been engaged continuously on the research side of the steel sheet and tinplate industry of South Wales.

P. M. Macnair

W. Hay

D. R. Brown, B.A.Eng.(Cantab.)—Blast-furnace engineer with Messrs. Ashmore, Benson, Pease and Co., Ltd., Stockton-on-Tees. Educated at Tonbridge School, Mr. Brown gained engineering experience in the shops of Ashmore, Benson, Pease and Co., before proceeding to Sidney Sussex College, Cambridge, where he graduated in engineering in 1937. On leaving Cambridge he joined the blast-furnace department of the Ford Motor Co., Ltd., Dagenham, and acquired a knowledge of the operation of a modern blast-furnace plant. During the recent war he served in the Royal Navy and in 1944 held the rank of lieutenant-commander. Mr. Brown rejoined Ashmore, Benson, Pease and Co. in 1946 as assistant to the Director in charge of their blast-furnace engineering work.

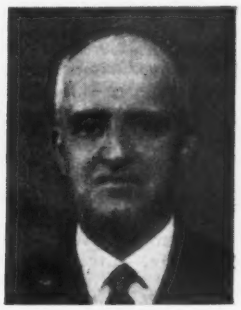
T. H. Stayman, A.M.Inst.C.E.—Chief Engineer of the McKee Iron and Steel Division of Messrs. Head, Wrightson and Co., Ltd. Mr. Stayman was educated at Stockton Grammar School and at Middlesbrough Technical High School. His apprenticeship was served with Messrs. Head, Wrightson as a draughtsman. After two years in the shops and four years as site engineer, Mr. Stayman became a designer and held that position until his present appointment in 1946.

J. Brimelow, B.Sc.(Eng.), M.Inst.C.E.—A technical advisor to Mitchell Engineering, Ltd., of London and Peterborough. Mr. Brimelow received his technical training at East London College and was awarded an honours degree in engineering in 1912. From 1912 to 1915 he served with Messrs. Fraser and Chalmers, Ltd., of Erith, and was then transferred to the Explosives Department of the Ministry of Munitions, as draughtsman

D. Petit

D. R. Brown

T. H. Stayman



J. Brimelow

and assistant engineer, where he remained until the end of the first World War. In 1918 he became assistant engineer to the National Metal and Chemical Bank, Ltd., and held this position until 1925 when he joined the staff of Mitchell Engineering, Ltd.

IRON AND STEEL ENGINEERS GROUP

Eighth Meeting

The Eighth Meeting of the Iron and Steel Engineers Group will be held on Wednesday, 24th November, 1948, at the Offices of the Institute, 4, Grosvenor Gardens, London, S.W.1., at 10.30 A.M. Papers on the handling of materials in iron and steel works will be presented and discussed. Further details of the Meeting will be found in the September, 1948, issue of the *Journal*.

NEWS OF SCIENCE AND INDUSTRY

Pageant of Production at Sheffield

At a Pageant of Production to be presented from 15th to 27th November, 1948, by the City of Sheffield, in the City Hall, Sheffield, the details of the industrial life and problems of the city will be shown by means of living isotypes.

During the fortnight of the Pageant the citizens and Civic Authority of Sheffield will arrange organized tours of the great industries of the city, including not only well-known iron and steel trades, but many examples of craftsmanship in other activities not usually associated with the city.

Further details of the Pageant may be obtained from the City Authorities or from the leading industrial concerns.

Dr. Ing. Carl Still

Dr. Ing. e.h. Carl Still, of Recklinghausen, attained his eightieth year on 2nd August, 1948. A reception was held by the Provost of Recklinghausen, and was attended by the Rectors of Münster and Göttingen Universities.

DIARY

9th–30th Oct.—ROYAL PHOTOGRAPHIC SOCIETY OF GREAT BRITAIN—93rd Annual Exhibition of Photography—Scientific, Nature, Record, and Technical Exhibits—16, Princes Gate, London, S.W.7.

21st–22nd Oct.—BRITISH IRON AND STEEL RESEARCH ASSOCIATION—Conference on Foundry Core Bonding Agents—Ashorne Hill, near Leamington Spa.

26th Oct.—LINCOLNSHIRE IRON AND STEEL INSTITUTE—Presidential Address, by H. M. Henderson.

4th Nov.—LEEDS METALLURGICAL SOCIETY—"*Some Recent Lines of Metallurgical Development*," by R. Genders—Chemistry Department, the University, Leeds—7.0 P.M.

5th Nov.—WEST OF SCOTLAND IRON AND STEEL INSTITUTE (Joint meeting with the Society of Chemical Industry, Glasgow Section)—"*Constitution–Property Relationships in Refractory Material*" by A. L. Roberts—Royal Technical College, Glasgow—7.15 P.M.

10th and 11th Nov.—THE IRON AND STEEL INSTITUTE—Autumn Meeting (*see* p. 213)—4, Grosvenor Gardens, London, S.W.1—10.0 A.M.

11th Nov.—LIVERPOOL METALLURGICAL SOCIETY (Joint Meeting with the Society of Chemical Industry)—"*A Review of Steels Used in the Chemical Industries*," by N. P. Inglis—The University, Brownlow Hill, Liverpool—7.0 P.M.

13th Nov.—SWANSEA AND DISTRICT METALLURGICAL SOCIETY—"*The Hardness and Deformation of Metals*," by Professor Hugh O'Neill—Royal Institution of South Wales, Swansea—6.30 P.M.

16th Nov.—EBBW VALE METALLURGICAL SOCIETY—"*Aluminium and Its Alloys*," by J. S. Walton—Lesser Workmen's Hall, Ebbw Vale—7.0 P.M.

16th Nov.—SHEFFIELD SOCIETY OF ENGINEERS AND METALLURGISTS (Joint meeting with the Sheffield Metallurgical Association and the Institute of Metals)—"*Plastic Working of Metals*," by E. Orowan—Sheffield Metallurgical Club, West Street, Sheffield—6.15 P.M.

19th Nov.—WEST OF SCOTLAND IRON AND STEEL INSTITUTE—"*Roll Founding*," by W. G. Scott—39, Elmbank Crescent, Glasgow—6.45 P.M.

23rd Nov.—LINCOLNSHIRE IRON AND STEEL INSTITUTE (Joint meeting with the Iron and Steel Engineers Group)—Paper by A. Bridge.

24th Nov.—IRON AND STEEL ENGINEERS GROUP—Eighth Meeting—4, Grosvenor Gardens, London, S.W.1—10.30 A.M.

TRANSLATION SERVICE

(The previous announcement was made in the September, 1948, issue of the *Journal*, p. 98.)

TRANSLATIONS AVAILABLE

No. 355 (German). "The Possibilities of Influencing the Oxidation Zone of the Blast-Furnace." (Report No. 227 of the Blast-Furnace Committee of the Verein Deutscher Eisenhüttenleute). (*Stahl und Eisen*, 1947, vol. 66–67, Aug. 14, pp. 277–284.)

No. 356 (French). M. QUILLATRE: "The Use of Wooden and Synthetic-Resin Bearings in Rolling Mills." (*Revue de Métallurgie, Mémoires*, 1941, vol. 38, Oct., pp. 253–271; Nov., pp. 285–298.)

No. 357 (German). G. NEUMANN: "How Can the Laws of Statics be Applied to the Design and Development of Furnace Roofs?" (*Archiv für das Eisenhüttenwesen*, 1942, vol. 15, Apr., pp. 437–445.)

No. 358 (German). A. EICHINGER: "Mathematical Treatment of Creep Curves." (*Archiv für das Eisenhüttenwesen*, 1940, vol. 13, Mar., pp. 397–402.

PAPERS ACCEPTED FOR TRANSLATION SINCE THE LAST ANNOUNCEMENT

(German). T. KOOTZ: "On the Question of the Effect of Converter Shape on the Nitrogen Content of Steel. (The 'Converter Characteristic.')" (*Verein deutscher Eisenhüttenleute, Düsseldorf, (Confidential Report No.* 72 (1944).)

(German) K. DAEVES and W. HOLTMANN: "The Behaviour of Nitrogen in Acid and Basic Side-Blown Converters." (*Verein deutscher Eisenhüttenleute, Düsseldorf, Confidential Report No.* 62 (1944).)

(German). O. KRIFKA and F. RAPATZ: "The Deoxidising and Desulphurising Slag-Reaction Process." (*Verein deutscher Eisenhüttenleute, Düsseldorf, Confidential Report No.* 77 (1944).)

The article by K. GUTHMANN in *Stahl und Eisen*, 1947, vol. 66–67, Dec. 4, pp. 425–427, announced in the September, 1948, issue of the *Journal*, has been retitled "The Rate of Driving of Blast-Furnaces."

TRANSLATIONS PREPARED AT MEMBERS' REQUEST—Members requiring translations of foreign papers are invited to communicate with the Secretary, who will ascertain whether the translations can be prepared for inclusion in the Series.

ABSTRACTS OF CURRENT LITERATURE *and* BOOK NOTICES

IRON AND STEEL MANUFACTURE AND RELATED SUBJECTS

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MINERAL RESOURCES

The Hematite of Labrador and New Quebec. W. M. Bonham. (Canadian Mining Journal, 1948, vol. 69, June, pp. 67–70). A review of the position reached in 1947 in the proving of the hematite deposits in Labrador and New Quebec is presented. Three deposits in Labrador are estimated to contain 41 million tons, and seven in New Quebec to contain 98 million tons. The area in which ore has been found is already as large as the Mesabi range. Railway and port facilities are being surveyed.

The Iron Sands of Fitzroy, New Plymouth. C. O. Hutton. (New Zealand Journal of Science and Technology, Section B, 1945, vol. 26, pp. 291–302). **Vanadium in the Taranaki Iron Ores.** C. O. Hutton. (New Zealand Journal of Science and Technology, Section B, 1945, vol. 27, pp. 15–16: Mineralogical Abstracts, 1948, vol. 10, Mar., pp. 222–223). Dunes on the coast contain a large amount of low-grade iron ore, from which it is estimated half a million tons of pure titanomagnetite could be extracted magnetically. Analysis by F. T. Seelye of the titanomagnetite gave Fe_2O_3 49·74%, FeO 29·46%, TiO_2 8·57%, Al_2O_3 2·49%, MnO 0·71%, MgO 2·81%, CaO 0·17%, V_2O_3 0·39%, P_2O_3 0·50%, Cr_2O_3 0·03%, SiO_2 2·95%, H_2O 0·38%, insoluble 2·03% = 100·23%. Other minerals present in smaller amounts are titanhematite containing Fe_2O_3 83·1%, FeO 5·58%, TiO_2 9·6%, Mg 1·5%, hornblende, apatite, zircon, etc. No vanadium mineral, *e.g.*, coulsonite, could be detected, and the vanadium is probably held in the magnetite lattice.

Contribution to the Mineralogic Study of the Oolitic Iron Ores of Lorraine (Briey Basin). S. Caillère and F. Kraut. (Bulletin de la Société Française de Mineralogie, 1946, vol. 69, Jan.–Dec., pp. 83–98).

The Iron and Manganese Ores of Gonzen. W. Epprecht. (Beitr. Geol. Schweiz. geotech. Ser. Lief. 1946, No. 24: [Abstract] Centre National de la Recherche Scientifique, Bulletin Analytique, 1948, vol. 9. No. 3/4, p. 660).

The Geology and Origin of Iron and Nickel Deposits near Křemže in Southern Bohemia. V. Čech and J. Koutek. (Sborník Státního Geol. Ústavu Československé Republiky, 1946, vol. 13, pp. 1–22).

A Review of Polish Ore Deposits from the Economic Viewpoint. R. Krajewski. (Hutnik, 1947, vol. 14, July–Aug., pp. 348–354). [In Polish]. A general review of Polish ore deposits is presented including the ores of iron, manganese, zinc, lead, copper, pyrites, nickel, arsenic, tin, cobalt, magnesite, barytes and fluorspar.—W. J. W.

New Iron Ore Deposits of the Karelo-Finnish S.S.R. V. I. Namoyushko and K. K. Khazanovich. (Gornyi Zhurnal, 1947, vol. 121, No. 5, pp. 3–7: Chemical Abstracts, 1948, vol. 42, July 10, cols. 4501–4502). Extensive ore deposits were discovered (1945–46) in the Lake Gimol' (?) region. The geology of these deposits is outlined. Preliminary determinations indicate in one part of the deposit a total iron content 35·29% and soluble iron 27·17%, the average sulphur content 0·23%, and phosphorus 0·13%. In another part of the deposit the total iron is 38·05–39·25% and the soluble iron is 35·01–35·48%. Generally the ore is similar to the iron ores of the Kola peninsula and of Scandinavia.

The Principal Metals and Ores of Norway. H. Bjørlykke. (Naturen, 1946, vol. 70, May, pp. 129–135). [In Norwegian].

Researches on the Iron Ores of Central Sweden between 1921 and 1946. P. Geijer. (Geologiska Föreningens i Stockholm Förhandlingar, 1946, vol. 68, Mar.–Apr., pp. 201–211).

Manganese Mineralization in the Ultevis District, Jokkmokk, North Sweden. Part I: Geology. O. H. Ödman. (Sveriges Geologiska Undersökning, Series 7, 1947, vol. 41, No. 487).

Report on Explorations of Coal Deposits in the Komló Area of Hungary in the Years 1936 to 1943. K. Telegdi-Roth. (Bányászati és Kohaszati Lapok, 1948, vol. 6, June, pp. 161–169). [In Hungarian]. Exploration work in the Komló area was started in 1936 and test borings were started the following year with deep-boring equipment suitable for bores to a depth of 800 m. The test bores were 407 to 874 m deep and details of the most important ones are given in the paper. The exploration work covered an area of 4,685,000 sq.m., and the average layer thickness of the coal deposits discovered was 10 m. The available coal deposits in this area are estimated at approximately 30 million tons and this is sufficient to permit an output of 600,000 tons/annum for 50 years.—E. G.

Titaniferous Magnetite Deposits of the Lake Sanford Area, New York. R. C. Stephenson. (Bulletin of the New York State Museum, Albany, 1945, No. 340).

ORES—MINING AND TREATMENT

Crushing Iron Ore. D. D. Howat. (Iron and Coal Trades Review, 1948, vol. 157, July 23, pp. 177–183; July 30, pp. 239–244). The theory and practice of crushing iron ore are comprehensively discussed. The first part deals with determining the efficiency of crushing machines, relation between useful work and size reduction, characteristics of fracture in brittle solids, conversion of energy into heat in crushing,

crushing machines and particle-size relationships, particle-size distribution and the design of crushing machines, and effect of size reduction on crusher characteristics. The second part, on the practical aspects, covers preliminary handling of run-of-mine ore, characteristics of primary-crushing machines, comparison of jaw and gyratory crushers, coarse-crushing rolls, differences between British and German practice in sledging rolls, single-roll crushers, stage crushing, secondary crusher, cone crusher, hammer mills, ore fines and sintering, and value of sinter in the blast-furnace charge.—R. A. R.

A Note on the Effects of Lime Additions to Iron Ore Sinters. J. M. McLeod. (Journal of the West of Scotland Iron and Steel Institute 1946–47, vol. 54, pp. 197–203). The poor reducibility of sinters is due to the coating of the particles of iron oxide with a ferrous silicate slag, of low porosity. The addition of lime displaces ferrous oxide from silicate compounds, rendering them more easily reducible as well as raising the melting point of the slag constituent, which therefore covers the micropores of the material to a lesser extent. Full-scale tests have shown that charges with 10% of lime can be fully sintered and that the time for 90% reduction is halved.—J. P. S.

Magnetic Concentration Experiments upon Iron Ores Used in North Lincolnshire Practice. L. Reeve. (Journal of The Iron and Steel Institute, 1948, vol. 159, July, pp. 275–280).

Ore Dressing, and Cement Machinery in Germany. (British Intelligence Objectives Sub-Committee, 1948, Final Report No. 1753 : H.M. Stationery Office). Descriptions are given of the following German ore and coal preparation machinery : (*a*) Double-cone impeller flotation machine ; (*b*) froth flotation machines for ores and coal ; (*c*) the Krupp subaeration flotation machine ; (*d*) Lurgi electrostatic separators ; (*e*) Starkfeld, Humboldt, Gröppel, and wet-drum magnetic separators ; and (*f*) vibratory ball mills. Cement kilns and machinery are also described.—R. A. R.

Concentratability of Birmingham, Ala., Red Iron Ores by Separation in Heavy Media. B. H. Clemmons, R. H. Stacy, and B. G. Saunders. (United States Bureau of Mines, 1948, Report of Investigations 4249). Samples of red iron ore from 10 of the 13 mines in the Birmingham, Ala. district, were concentrated by separation in heavy media of 2·90, 3·00, and 3·20 specific gravities. The relatively porous surface of weathered red ores did not respond to the treatment, but ore from four of the mines did respond quite well. With a daily production of about 12,000 tons from these four mines a theoretical rejection of 1255 tons of shale and ferruginous sandstone was calculated from the test data.—R. A. R.

The Preparation of Iron Ore for Blast Furnace and Open Hearth Use. R. R. Williams, jun. (American Iron and Steel Institute, May 26–27, 1948, Preprint). The ore bedding, crushing, screening and sintering equipment and processes of the Colorado Fuel and Iron Corp. are described.—R. A. R.

REFRACTORY MATERIALS

Refractory Materials and Metallurgy. Louise Halm. (Revue Technique Luxembourgeoise, 1948, vol. 40, July–Sept., pp. 108–125). The manufacture, properties, and metallurgical applications of refractory materials are surveyed. R. F. F.

A Note on the Manufacture of Silica Bricks in Canada. N. F. Dufty. (Refractories Journal, 1948, vol. 24, July, pp. 248–249). A brief account is given of the process employed by the Algoma Steel Corporation, Sault Ste. Marie, Canada, for making silica bricks using pure quartzite from a deposit only 20 miles away.—R. A. R.

Dutch Report on Visit to Refractory Brick Factories in the British Zone of Germany. (British Intelligence Objectives Sub-Committee, 1948, Miscellaneous Report No. 78 : H.M. Stationery Office).

Dutch Report on Visit to Refractory Brick Factories in the American and French Zones of Germany. (British Intelligence Objectives Sub-Committee, 1948, Miscellaneous Report No. 82 : H.M. Stationery Office).

Production of Dinas at the Bad-Lausigg Plant (Germany). P. I. Galkin. (Ogneupory, 1946, vol. 11, No. 9–10, pp. 10–17 : American Ceramic Abstracts, 1948, June 1, p. 131). Details of the manufacture of Dinas at the Bad-Lausigg plant in the Soviet occupation zone of Germany are given. The plant has an annual output of 30,000 tons of Dinas, shapes used in steel-melting furnaces, gasworks and coke-oven batteries. The chief raw material is quartzite containing 96·30–97·3% SiO_2, 1·9–3·70% Al_2O_3 + TiO_2, 0·26–0·6% Fe_2O_3, and up to 0·54% CaO. Special attention is given to maintaining the specific gravity of the milk of lime at 1·3 ; that of the sulphite liquor is kept at 1·25. For each 500 kg of batch, 35 l. of milk of lime and 2 l. of sulphite liquor are used.

Firing of Dinas in Round Periodic Kilns. M. B. Gotlib. (Ogneupory, 1946, vol. 11, No. 11–12, pp. 3–13 : American Ceramic Abstracts, 1948, June 1, p. 129). Experiments conducted over a period of one and a half years at the Stalin Metallurgical Works at Magnitogorsk show that it is possible to fire and cool Dinas in round periodic kilns of 180 cu.m. capacity in 12 to 12½ days. Further acceleration is possible. The following firing and cooling cycle is proposed :

Temp. Interval, °C	Rate, °C/hr	Duration, hr
400–600	20	28
600–1200	25	24
1200–1350	10	15
1350–1460	3	37
1460	0	40
1460–400	35	30
400–250	10	15
250–50	5	40

Dinas Brick with Magnesia Binder. V. A. Bron. (Comptes Rendus [Doklady] de l'Académie des Sciences de l'U.R.S.S., 1948, vol. 59, No. 2, pp. 285–287 : Fuel Abstracts, 1948, vol. 3, June, p. 103). The author shows by means of tables that additions of 1·78% to 4·59% of magnesium oxide to Dinas brick improve its refractory properties under working conditions in open-hearth furnaces.

Ceramic Properties of Finely Dispersed (Wet Ground) Andalusite. D. N. Poluboyarinov and R. Ya. Popil'skii. (Ogneupory, 1946, vol. 11, No. 9–10, pp. 35–43 : American Ceramic Abstracts, 1948, June 1, p. 129). Andalusite refractories made in the Soviet Union have a clay binder which it is proposed to replace with finely ground andalusite. Compared with clay binder, the andalusite binder raised the temperature of initial deformation under load by 90° C. although the increase in alumina content was only 4%. Compressive strength was also increased, and porosity was improved. In connection with this work, sillimanite refractories imported from the United States and intended for use in " Detroit " arc furnaces were tested. Results shows that a very small amount of clay or none at all was used as a binder.

Effect of Gas Atmosphere on Furnace Heat Losses. G. C. Nonken. (Steel Processing, 1948, vol. 34, July, pp. 377–381). A method for determining the thermal conductivity of refractories the pores of which were filled with air, hydrogen, nitrogen, and a prepared atmosphere is described. This was applied to firebrick, insulating firebrick, a refractory concrete, and fused magnesium oxide powder, and the results obtained at different temperatures are presented by curves. The results for hydrogen were always much higher than those for the other gases tested which had approximately the same conductivities. H. W. Russell's method of calculating the conductivity of porous insulators with two different pore-and-solid arrangements are quoted and used to calculate the heat loss of a medium-sized tool-room furnace.—R. A. R.

The Testing of Silica Bricks for Service in Open-Hearth Furnace Roofs. C. H. Bacon. (Transactions of the British Ceramic Society, 1948, vol. 47, July, pp. 233–258). An attempt has been made to correlate laboratory tests on sample bricks with the service obtained in the furnace. The maintained-load test is considered the most important single test, poor results from this being associated with high specific gravity or high alumina content. Transverse crushing strength is also regarded as important, and is connected with low specific gravity. Photographs of furnace roofs are shown, which record the nature of wear and the stage in the campaign at which it develops. —J. P. S.

The Calculation of Heat Transfer through Two-Media Furnace Walls. Agnes H. Waddell. (Journal of the West of Scotland Iron and Steel Institute, 1946–47, vol. 54, pp. 205–220). Suggestions are made for the graphical presentation of analytical results concerned with the flow of heat through furnace walls composed of a layer of firebrick in contact with a layer of insulating brick, by the use of the dimensionless

quantities thermal-resistance ratio, "relative time" ratio, position ratio, temperature-difference ratio, and the ratio of the thermal conductivities of the two media.—J. P. S.

Investigation of the Sintering of Magnesia with the Electron Microscope. F. Kahler. (Radex Rundschau, 1947, Oct., pp. 50–55). The residues after the removal of CO_2 at temperatures between 400° and 1000° C from preparations made from natural magnesium carbonate were examined with the electron microscope. A loose pseudomorphous structure of magnesia particles of ultra-microscopic size is left at 400° C, which knits with increasing temperature into a larger secondary aggregate with increasing porosity. The first sign of sintering with a marked increase in porosity takes place at 900° C, with definite sintering at 1000° C at which temperature the secondary grains scarcely exceed 5μ. Depending on the composition, further agglomeration into macroscopic dimensions takes place at higher temperatures—R. A. R.

The Chemical and Mineralogical Structure of Refractory Magnesite Aggregates. K. Konopicky and F. Trojer. (Radex Rundschau, 1947, June, pp. 3–15). The published results of investigations of phases and the construction of equilibrium diagrams for the systems pertaining to magnesite refractories are presented and reviewed, and in some cases amplified, and the separate minerals which may occur are described from the chemical and mineralogical points of view, microfilms of the structures being reproduced.—R. A. R.

Factors in Service Behaviour of Silica Brick. L. A. Smith. (American Iron and Steel Institute, 1948, May 26–27, Preprint: Blast Furnace and Steel Plant, 1948, vol. 36, June, pp. 701–706). The properties of the raw materials for, and the manufacture of, silica bricks are discussed with notes on open-hearth furnace design, heating up a new furnace lining, and the importance of careful furnace control and good management.—R. A. R.

The Suspended Basic Roof. L. Hütter. (Radex Rundschau, 1947, June, pp. 16–23). The requirements of the all-basic open-hearth furnace lining are stated, data are presented on the life of a large number of Radex-lined furnaces, and full details of the brick shapes and methods of suspension with copies of two standard specifications are given.—R. A. R.

Basic Brick in the Open-Hearth Furnace. V. W. Jones. (American Iron and Steel Institute, May 26–27, 1948, Preprint). An account is given of the results obtained with basic bricks in open-hearth furnaces, in particular basic and silica brick construction and maintenance for the furnace ends are compared. In general, maintenance and fuel costs are lower with basic ends.—R. A. R.

Properties and Performance of Open Hearth Bottoms. H. M. Kraner. (American Iron and Steel Institute, May 26–27, 1948, Preprint). Reactions between slags and open-hearth furnace bottom refractories are considered and the behaviour of rammed and sintered magnesite bottoms is studied by reference to the analysis at increasing depths of cores drilled from the hearth at various positions.—R. A. R.

Modern Refractories for Open Hearth Furnace Bottoms Highly Resistant to Hydration. (Industrial Heating, 1948, vol. 15, May, pp. 842–846). *See* Journ. I. and S.I., 1948, vol. 159, Aug., p. 433.

A New Type of Lining for Rotary and Cylindrical Furnaces. R. Maurier. (Radex Rundschau, 1947, June, pp. 29–30). The advantages of using rhomboid-shaped bricks for the linings of rotary kilns and cylindrical furnaces are explained. R. A. R.

Improving the Properties of Grog Products Suitable for Lining the Shafts of Blast Furnaces. E. K. Keler and V. N. Abramova. (Ogneupory, 1946, vol. 11, No. 4–5, pp. 18–30: American Ceramic Abstracts, 1948, May 1, pp. 102–103). Brief accounts are given of research activities at the Leningrad Institute of Refractories to develop methods of testing the abrasion and impact resistance of blast-furnace refractories. The effects of existing Russian methods of manufacture on these two characteristics were also investigated. Two variations of brick manufacture are outlined. One method utilizes Chasov-Yar clay PK to produce brick suitable for lining the upper two-thirds section of the shaft, and the other method uses Latninsk clay (Grade I) for the lower one-third section. The specifications for brick for the lower section are as follows: (*a*) $Al_2O_3 + TiO_2$ not lower than 39%, (*b*) refractoriness not below 1730° C, (*c*) volume porosity not over 12%, (*d*) gas absorption not over 0·2%, (*e*) compressive strength not less than 400 kg/sq.cm., and (*f*) abrasion (Bauschinger) not over 0·10 g/cu.m.; for brick of the upper section the respective values were (*a*) not lower than 24%, (*b*) not below 1630° C, (*c*) not over 10%, (*d*) not over 0·2%, (*e*) not less than 400 kg/sq.cm. and (*f*) not over 0·10 g/cm.

Refractories for Boiler Furnaces. (Engineering and Boiler House Review, 1948, vol. 63, May, pp. 132–135). The effects of temperatures and attack by combustion products on silica and semi-silica bricks, firebrick, and alumina brick boiler refractories under operating conditions, are explained. The bonding of bricks and choice of a suitable jointing cement are discussed.—M. A. V.

Refractory Materials for Reheating Furnaces. H. Parnham. (Iron and Coal Trades Review, 1948, vol. 156, June 11, pp. 1215–1220). The chemical erosion, mechanical abrasion, and spalling of refractory hearth materials in reheating furnaces are reviewed with special reference to the melting points of systems containing FeO. Refractories available for hearths and their properties are listed, and the advantages of plastic chrome known as "Plastic K-N" are pointed out with examples of its use in several types of large and small reheating furnaces.—R. A. R.

The Lining of Melting Furnaces and the Quality of Steel. F. Rapatz. (Radex Rundschau, 1948, Apr., pp. 9–14). The nature and effects of inclusions in steel arising from basic and acid furnace linings are discussed. In general, inclusions from a basic lining are less harmful than those from an acid lining. At rolling temperature the latter are plastic and glossy and tend to layer formation whereas the former are broken up by the rolling. Inclusions which melt at rolling temperature give rise to red-short conditions.—R. A. R.

BLAST-FURNACE PRACTICE AND THE PRODUCTION OF PIG IRON

Blast-Furnace Stoves. J. E. MacDonald and J. P. Marron. (Iron and Steel, 1948, vol. 21, June, pp. 295–296). To obtain greater efficiency in blast-furnace stoves without rebuilding them to a greater diameter (a course not possible for lack of room and time) it is proposed to modify existing stoves by the construction of off-set combustion chambers outside the chequer work but within the same steel shell; this will add 25% to the chequer volume and it can be carried out during normal stove relining.—J. P. S.

Sinter Quality and Effect of Sinter on Blast Furnace Practice. J. L. Mauthe. (Eastern States Blast Furnace and Coke Oven Association: Blast Furnace and Steel Plant, 1948, vol. 36, July, pp. 817–824). Through close control of sintering operations, particularly of particle size, in the sinter, and its porosity, and avoidance of excessive temperatures liable to produce a coating of iron silicate on the particles, increased iron output from the blast-furnace, combined with reduced coke consumption and less flue dust, can be produced. J. P. S.

Considerations in the Use of Carbon Refractories in the Blast Furnace. W. S. Debenham. (American Iron and Steel Institute, May 26–27, 1948, Preprint: Steel, 1948, vol. 122, July 12, pp. 110, 113, 124–130). Experience in the use of carbon refractories for blast-furnace hearths and linings in Germany, England, and the U.S.A. is reviewed, and, on the basis of this, a design for lining a hearth 25 ft. in dia. is suggested.—R. A. R.

Note on the Blowing of Hydrogen in the Blast-Furnace. J. G. Platon. (Revue de Métallurgie, Mémoires, 1948, vol. 45, Mar.–Apr., pp. 118–119). The weight of hydrogen required to combine with a given weight of oxygen in an iron ore is much less than the corresponding amount of carbon (as such or as CO). Also reduction by hydrogen is more exothermic than that by carbon, whether direct or indirect. While hydrogen itself is likely to be too expensive for use in the blast-furnace for many years to come, natural hydrocarbons may not be; the author presents the calculation of a materials and thermal balance for the use in the blast-furnace of a hydrocarbon containing 15% of hydrogen and 85% of carbon. He concludes that half the hydrogen would be recovered in the gas, which would be enriched, that 80 kg of fuel oil would replace 130 kg of coke, or, allowing for the greater latent heat of the blast-furnace gas, 80 kg of fuel would be equivalent to 160 kg of coke. The introduction of fuel and of hydrocarbons into the hearth of the blast-furnace presents difficulties, but it is at present possible to

introduce 100–200 kg of fuel or pulverized coal per ton of iron, with a direct economy of 250–300 kg of coke.—A. E. C.

Desulphurization and Blast-Furnace Slags. A. K. Prestud. (Tidsskrift for Kjemi, Bergvesen og Metallurgi, 1947, vol. 7, Jan., pp. 4–11 : [Abstract] Centre National de la Recherche Scientifique, Bulletin Analytique, 1948, vol. 9, No. 5–6, p. 953). An account is given of the part played by the slag in the desulphurization of iron in both electric and ordinary blast-furnace operation. The influence of the viscosity and temperature of the slag is discussed and reference is made to the formulae of Holbrook and Joseph.

Elimination of Sulphur in the Blast-Furnace. D. Joyce. (Journal of The Iron and Steel Institute, 1948, vol. 159, July, pp. 291–296).

The Effect of Operating Conditions on Type of Reduction and Carbon Rates in the Blast Furnace. J. Taylor. (Journal of the West of Scotland Iron and Steel Institute, 1946–47, vol. 54, pp. 137–176). It is shown that for any given set of conditions in the blast-furnace, the maximum efficiency is associated with the maximum proportion of indirect reduction, according to the reaction

$$Fe_2O_3 + 9C + 4\tfrac{1}{2}O_2 \rightleftharpoons 2Fe + 3CO_2 + 6CO + 243{\cdot}7 \text{ cal}$$

rather than direct reduction

$$Fe_2O_3 + 2\tfrac{1}{4}C \rightleftharpoons 2Fe + \tfrac{3}{4}CO_2 + 1\tfrac{1}{4}CO - 88 \text{ cal}$$

This maximum varies with slag bulk, blast temperature, radiation losses and the moisture content of the burden, but only by attaining it can the minimum CO/CO_2 ratio in the top-gas, and maximum efficiency be attained. It is not, however, easy in practice to discover when too much direct reduction is occurring, though it may be determined from studying the effects of coke rate increase on the top-gas CO/CO_2 ratio.—J. P. S.

DIRECT PROCESSES

Direct Reduction of Iron Ore with Charcoal. F. A. de T. Piza. (Boletim da Associação Brasileira de Metais, 1947, vol. 3, pp. 490–495 : Instituto de Pesquisas Tecnológicas, Separata No. 180 : Chemical Abstracts, 1948, vol. 42, May 10, col. 2899). Tests indicated that when pig iron is expensive and scrap is scarce direct reduction of pure iron ores with wood charcoal may be economical, and pilot-plant study for Brazil is recommended. Ceramic or high-temperature alloy pots are practical. Two hours at 1000° is insufficient and twelve hours sufficient to reduce completely 16-kg charges.

Manufacture of Sponge Iron in Ceramic Tunnel Kilns. V. H. Gottschalk, E. S. Beebe, and R. S. Cole. (United States Bureau of Mines, 1948, Report of Investigations 4271). A report is presented on the production of sponge iron in rectangular crucibles 24 in high which were charged on cars in a tunnel kiln 200 ft long, 56 in wide, and 60 in high, fired with fuel oil and coal. It was established that kilns as used for refractory ceramic ware are satisfactory for making sponge iron. The degree of reduction and the sulphur content of the product can be controlled satisfactorily.—R. A. R.

Experience to Date on Iron Production by Methods other than Cold Blast Furnace : Direct Reduction. E. C. Smith. (American Iron and Steel Institute, May 26–27, 1948, Preprint). Descriptions are given of direct processes for the production of iron by methods other than the coke and charcoal blast-furnaces. The Chinese "Shansi" method, and the Höganäs, Wilberg, and Krupp-Renn processes are considered as economically successful, and a number of uneconomic processes and trials are reviewed with references to the literature and patents.—R. A. R.

PRODUCTION OF STEEL

Recent Developments in the Steel Industry. C. H. Desch. (Royal Institution of Great Britain, Nov. 21, 1947). After pointing out the effects of war on the steel industry, many advances in processes of manufacture and properties were referred to with particular reference to steels for use at high temperatures and magnet steels. Additional developments dealt with were related to methods of scientific control and research such as the quick-immersion thermocouple, the electron microscope, microchemical analysis, and detecting cracks in steel by supersonic methods.—R. A. R.

Progress in Iron and Steel Metallurgy in Recent Years. J. Gørrissen. (Tidsskrift for Kjemi, Bergvesen og Metallurgi, 1947, vol. 7, May, pp. 63–71).

Steel in Australia. (British Steelmaker, 1948, vol. 14, July, pp. 325–327). An account is given of the development of the steelworks of the Broken Hill Proprietary Co., Ltd. The total ingot capacity is now 1,900,000 tons/annum. It is planned to erect a hot and cold wide-strip mill and tinplate plant adjacent to the Port Kembla Works.—R. A. R.

The Brunswick Works of the Reichswerke A.G. and Operational Results for the Years 1942 to 1944. K. Hofmann and E. Peetz. (Stahl und Eisen, 1948, vol. 68, June 17, pp. 213–228 ; July 15, pp. 255–268). A comprehensive account is given of the plant, practice, and achievements in the period 1942 to 1944 of the integrated iron and steel works of the Reichswerke A.G. at Brunswick. The acid-smelting process is employed for the siliceous Salzgitter ore. In 1944 coke consumption was 1166 kg per 1000 kg of pig iron. Spherical 50-ton converters were tried but with wear of the lining the ratio of maximum diameter to blowing area became too high and the narrower design had to be reintroduced. Tilting open-hearth furnaces of 120–180 tons and arc furnaces of 40–60 tons were employed in conjunction with the converters. In the rolling mill there was a 6-stand billet mill consisting of three horizontal stands alternating with three vertical stands. Operating data on many processes are given in 30 tables.—R. A. R.

Investigations on Converter Shape and Blowing Conditions for Producing Steel Low in Nitrogen. T. Kootz and G. Gille. (Stahl und Eisen, 1948, vol. 68, Aug. 12, pp. 287–294). Investigations to improve the design of converters are reported. A horizontal barrel converter 2·5 m inside dia and 5 m long with a single line of tuyeres parallel to the axis was tried ; it produced steel very low in nitrogen but had very poor blowing properties. Tests were made with model converters with glass ends, and wave formation with different depths of charge, different blowing angles and tilting angles were studied and the results were applied to the design of full-scale upright converters. The optimum shape was that of a double-bellied converter with the tuyeres arranged for oblique blowing and shallow immersion. The depths of molten metal controls wave formation and the losses by ejection, and this depth increases with the amount of lime and slag drawn into the metal circulating in the converter. Lime coated with molten slag can be wetted and is more easily drawn into the moving liquid iron than unwetted lime which floats and accumulates round the converter sides and tends to damp wave formation.

Tests made by blowing an ammonia–air mixture in model converters indicated that wave formation by the melt did not affect the amount of nitrogen absorbed.—R. A. R.

Tonnage Oxygen for Increased Iron and Steel Production. J. H. Strassburger. (American Iron and Steel Institute, May 26–27, 1948, Preprint). Results achieved in the use of oxygen-enriched blast in the blast-furnace, Bessemer converter and open-hearth furnace at the works of the Weirton Steel Co. are reported and discussed, and the oxygen plant now under construction to produce 400 tons/day of 93–95%-purity oxygen is described.—R. A. R.

Development of an Improved Basic Bessemer Steel. H. A. Dickie. (Journal of The Iron and Steel Institute, 1948, vol. 159, Aug., pp. 360–375).

Multiple Fuel Burners for Open-Hearth Furnaces. J. M. Brashear. (Iron and Steel Engineer, 1948, vol. 25, May, pp. 60–68). Descriptions are given of designs of open-hearth burners for using oil atomized with steam, atomized oil with coke-oven gas,and atomized oil with coke-oven gas and oxygen. Operating data for 100-ton and 175-ton furnaces are given. With steam for atomizing, its volume and temperature are more important than the pressure. Maximum flame temperatures are obtained with ratios of 2·6–3·1 lb of steam per U.S. gallon of oil or tar, and 2·9–3·3 lb of steam per U.S. gallon of pitch. Keeping the same steam rate when the oil rate is reduced causes the flame temperature to fall considerably. The above ratios also apply when 35% of the total fuel is coke-oven gas.—R. A. R.

Open Hearth Furnace Models. A. R. Philip. (Nature, 1948, vol. 162, July 31, p. 188). In a water model of an open-hearth furnace it is found that the introduction of a stream of gas and illumination by means of a discharge tube enables the flow to be studied in highly turbulent conditions.—J. P. S.

The Modern Open Hearth. T. J. Ess. (Iron and Steel Engineer, 1948, vol. 25, July, pp. O-19–O-70). A comprehensive description is given of the modern open-hearth melting

shop with details and illustrations of the separate items of plant and equipment. Data on the analyses and consumption of materials and fuels are presented in 22 tables.—R. A. R.

Utilizing Cupola Metal in Open Hearth Charge. A. W. Gregg. (Steel, 1948, vol. 123, July 19, pp. 116–121). The layout of plant and equipment for operating two cupolas alternatively to supply hot metal for charging into open-hearth furnaces is described. The cupolas are operated for 72-hr periods between repairs.—R. A. R.

Some Practical Aspects of Direct Oxidation in the Basic Open Hearth Process. F. G. Norris and E. B. Hughes. (American Iron and Steel Institute, May 26–27, 1948, Preprint). Results obtained by the Wheeling Steel Corp. when blowing oxygen into the bath in the basic open-hearth furnace are recorded on the basis of rate of removal of carbon, reduced heat time, and decrease in skull formation.—R. A. R.

Application of Oxygen to Open-Hearth Furnaces. D. D. Howat. (Iron and Coal Trades Review, 1948, vol. 156, June 25, pp. 1337–1343; vol. 157, July 2, pp. 7–11). The author discusses recent American work on this subject, and on the basis of recent visits to a number of plants in America and Canada, concludes that there are many advantages to be gained in the use of oxygen in combustion for melting down, and for carbon removal. The economic considerations are still very important, and the most useful point of application remains controversial; in low carbon heats the value of oxygen is greatest when used for refining, but in higher carbon heats its use in combustion or directly on the scrap may be more important in view of the time saved.—J. P. S.

Oxygen in the Electric Furnace. J. H. Berryman and J. M. Crockett. (Iron Age, 1948, vol. 162, Aug. 5, pp. 72–75, 144). Oxygen can be used in the basic electric-arc furnace for decarburization, either by means of the "oxygen lance" or by the direction of a strong blast of the gas through the slag cover; it is also useful for melting scrap remaining on the bottom, and for raising the temperature and promoting general bath action. In the smaller, acid electric-arc furnaces as used in steel foundries, considerations are similar, although high pressures should not be used for fear of damage to the roof refractories; even where it is possible to use ore for rapid carbon removal, oxygen has its use in bringing the bath up to the temperature at which this takes place most rapidly. Where the use of ore produces too violent a reaction oxygen may be used, as its action, though equally rapid, occurs more uniformly.—J. P. S.

Use of Oxygen for Decarburization and Melting in Electric Furnaces. J. H. Eisaman. (American Iron and Steel Institute, May 26–27, 1948, Preprint). Experience in the use of oxygen in electric furnaces making stainless steel at the Carnegie-Illinois Steel Corp. by the Chelius patent process is recorded. Oxygen injection has been successful and has completely replaced ore and scale.—R. A. R.

Control of Maximum Demand in Steel Works. (Engineering, 1948, vol. 166, Aug. 20, pp. 176–177). The electric melting shops of Firth Brown Steels, Ltd., Sheffield, are supplied with power from public mains under certain conditions which provide, *inter alia*, that the maximum demand at certain times shall not exceed 22 MW in winter and 28 MW in summer. The five electric-arc furnaces have therefore been provided with a load-indicating system which is linked with the printometer equipment in such a way that the actual and permissible demands can be compared at any time. The instruments for doing this are described and illustrated.—R. A. R.

Effect of Acid Oxides upon Desulfurization in the Iron–Manganese System. M. B. Royer and A. J. Williams, jun. (United States Bureau of Mines, 1948, June, Report of Investigations 4300). The mechanism of the desulphurization of molten manganese and molten iron with acid slags was studied. It was found that acid oxides are not desulphurizing agents in their own right. Interaction occurs between the acidic constituents of the slag and components of the metal which adjusts the slag composition to a state which is effective for desulphurizing. Thus, the acid oxides function only to the extent that their presence is required to form slag–mineral combinations having the requisite fluidity for dissolving sulphides. Sulphur can be removed by slags which contain no silica.—R. A. R.

Mechanism of Nozzle Erosion in Open-Hearth Ladles. R. B. Snow and J. A. Shea. (Steel, 1948, vol. 123, July 26, pp. 88, 91, 94). A number of tests has shown that nozzles of increasing refractoriness (expressed as pyrometric cone equivalent) are attacked to a decreasing extent by fine-grained aluminium-killed steel, but to an increasing extent by rimming steel and higher manganese coarse-grained steel. This latter effect is believed to be due to the presence of pores in the more refractory nozzles into which the FeO- and the MnO-bearing liquid penetrates.—J. P. S.

Slab and Billet Conditioning Symposium—Pneumatic Hand Chipping. P. S. Paluch. (Iron and Steel Engineer, 1948, vol. 25, June, pp. 86–87). The tools and technique for conditioning billets with pneumatic hammers are described. R. A. R.

Slab and Billet Conditioning Symposium—Application of Hand Scarfing to Modern Surface Conditioning. P. S. Paluch. (Iron and Steel Engineer, 1948, vol. 25, June, pp. 87–91). Scarfing billets with the oxy-acetylene torch is described and data on oxygen consumption and scarfing speeds are given.—R. A. R.

Slab and Billet Conditioning Symposium—Billet Conditioning with Mechanical Chippers. J. H. Vollmer. (Iron and Steel Engineer, 1948, vol. 25, June, pp. 91–97). Statistical analysis of the costs of conditioning 25,000 tons of billets has revealed that machine chipping is the most economical method; hand scarfing costs 63% more, and hand chipping 125% more. A chipping machine weighing 45 tons is described and illustrated. This machine has a cutting head carrying eight high-speed steel tools. The head is mounted on an arbor which can move across the billet; this movement, plus the vertical movement of the elevator and the forward and reverse travel of the carriage, all of which are electrically controlled, provide all the action necessary for light or heavy cutting at the desired positions.—R. A. R.

Steel Casting-Pit Practice and Ingot Defects. R. N. Duncan. (West of Scotland Iron and Steel Institute: Engineering, 1948, vol. 166, July 30, pp. 117–118). *See* Journ. I. and S.I., 1948, vol. 159, May, p. 87.

The Application of Statistical Methods to the Study of Ingot Cracking. I. M. Mackenzie and T. Urie. (Journal of the West of Scotland Iron and Steel Institute, 1946–47, vol. 54, pp. 177–196). The problem of the variation in cracking of ingots is complex and can only be treated by the use of statistical concepts suited to a system in which there are a large number of important variables, several of which have no numerical values. The author describes such a method, taking into account the factors of metal composition, slag composition, method of working the charge, and mould condition.—J. P. S.

PRODUCTION OF FERRO-ALLOYS

A Survey of Various Electric Smelting Processes and Their Application to India. H. Christiansen. (Current Science, 1947, vol. 16, Sept., pp. 269–273: Electrical Engineering Abstracts, 1948, vol. 51, May, p. 130). The processes described are for the production of FeSi, CuC, FeCr, FeMn, and pig iron. Operating data are included and typical furnaces illustrated (15-MW carbide, 10-MVA ferro-alloy, 6-MW rotating-hearth, 7·5-MVA Tysland-Hole). The principle of the electric smelting furnace is shown by a section through a smelting plant. The Söderberg continuous self-baking electrode is generally used. The electrometallurgical field in India has great possibilities owing to her abundant mineral resources and the large hydro-electric power plants which are being planned.

The Production of Ferro-Alloys in Germany during the Years 1939 to 1946. G. Volkert. (Stahl und Eisen, 1948, vol. 68, July 15, pp. 268–271). Progress in Germany in the methods of producing the ferro-alloys containing chromium, manganese, molybdenum, silicon, titanium, vanadium, tungsten, and zirconium-silicon, is reported.—R. A. R.

Investigations of the Thermite Process for the Production of Ferrotitanium. K. Giesen and W. Dautzenberg. (Stahl und Eisen, 1948, vol. 68, Apr. 22, pp. 159–162). Laboratory and full-scale tests for the production of ferrotitanium by the reduction of Norwegian ilmenite with thermite are described. The mixture to be reduced must be preheated to 300–450° C. The aluminium addition should be about 94% of that theoretically required, and the grain size of the aluminium should be 0·2–0·5 mm. An addition of lime equal to 20% of the aluminium helps the reaction and forms a thin slag out of which the

TiO_2 is driven. Ferrotitanium with 27% of titanium can be produced from unprepared ilmenite containing 44% of TiO_2.—R. A. R.

FOUNDRY PRACTICE

Steel Foundry Control. G. A. Lilliequist. (Transactions of the American Foundrymen's Association, 1946, vol. 54, pp. 553–567). The author discusses control of steel melting, sand processing and testing, heat-treatment, and inspection in the production of steel castings. Slag control, temperature control, fluidity tests, deoxidation control, inclusions in steel, metallographic examination, and physical testing are touched upon.—R. E.

Steel Castings for Aircraft. E. J. Brown and F. Rodgers. (Institute of British Foundrymen, June 8–11, 1948, Paper No. 902). This paper gives an account of the production of steel castings for aircraft parts, such as rear spars and pivot brackets, which are not stressed in flight. The factors affecting the choice of steels, the method of casting, and subsequent heat-treatment are discussed and the results of tests are detailed.—J. C. R.

New Techniques in a Small Steel Foundry. (Iron and Coal Trades Review, 1948, vol. 157, July 16, pp. 123–130). A description is given of the plant, layout, and processes at the steel foundry of Catton and Co., Ltd., Leeds. The melting plant consists of two 2-ton side-blown converters each with two spare bodies to which metal is supplied from two cupolas. Oxygen is added to the blast at 25 lb/sq.in. on the outlet side of the blower, the addition being 10% ; the total volume of air and oxygen is kept constant at 2000 cu.ft./min at a pressure of $2\frac{1}{2}$ lb/sq.in. The effects of oxygen enrichment on blowing time, blowing loss, bath temperature, and carbon, silicon, and manganese contents are shown by curves. The melting procedure, sand preparation, heat-treatment, and testing are also described.—R. A. R.

The Application of Oxygen Enrichment to Side-Blown-Converter Practice. J. L. Harrison, W. C. Newell, and A. Hartley. (Journal of The Iron and Steel Institute, 1948, vol. 159, July, pp. 281–290).

The Design of Steel Castings. G. Kniaginin. (Hutnik, 1947, vol. 14, Sept., pp. 407–413). [In Polish]. The design of steel castings is discussed ; the provision of adequate feeder heads and the importance of fillets at angles are illustrated with several examples of heavy castings.—W. J. W.

The Cleaning of Steel Castings. A. B. Lloyd. (Institute of British Foundrymen, June 8–11, 1948, Paper No. 903). An account is given of the various processes for cleaning steel castings. Particular reference is made to the choice of moulding materials in view of their effect on subsequent fettling operations.—J. C. R.

The Future Development of the Iron Foundries in the Netherlands. P. Schoenmaker and F. G. van Riet. (Metalen, 1948, vol. 2, Aug., pp. 259–262). [In Dutch].

Problems of Contraction and Distortion in Cast Iron Castings. E. Longden. (Institute of British Foundrymen, June 8–11, 1948, Paper No. 907 : American Foundrymen's Association, May 3–7, 1948, Preprint No. 48-10). This paper constitutes an addendum to a previous paper on the contraction of castings (*see* Journ. I. and S.I., 1940, No. II, p. 98A). Data on the differential contraction of different sections in large iron castings are presented and discussed and general rules for the contraction are deduced. Thicker members of a single casting contract more than thinner members, but if the same sections are cast separately, the reverse takes place. The thin sections of a single casting are subjected to extensional stress as the metal cools, created by the thicker sections which are not yet contracting. A study of the arrest and expansion periods on the cooling curves indicates that a thick section may be expanding at a time when a thinner section has passed the expansion phase and is contracting ; the thin sections will then suffer extensional stress and thick sections compressive stress. Heavy sections must take up a shorter length on cooling by bending certain parts of the casting, or by fracture of the weakened or most highly stressed sections.

Examples are shown of the introduction of an air-cooling system to zones of the casting with a high heat content ; this reduces the stresses and removes the need for high camber allowances.

A chart from which the camber allowance for castings of any length, depth, and wall section, can be readily ascertained, is presented.—R. A. R.

Application of the Brinell Test in the Foundry. J. Leonard. (Institute of British Foundrymen : Foundry Trade Journal, 1948, vol. 85, July 1, pp. 3–5). Illustrating his argument with hardness measurements taken on cast-iron press frames and cast on test-bars, the author, without wishing to draw definite conclusions, gives his opinion that Brinell tests may give some useful information about the regularity of castings. J. P. S.

The Use of Basic Grade Pig Iron in Iron Founding. J. E. Rehder. (Canadian Metals and Metallurgical Industries, 1948, vol. 11, June, pp. 24, 30). Recommendations are made on the melting practice to adopt in the foundry when basic iron has to be used owing to the shortage of foundry iron. Data on the necessary additions of ferrosilicon and ferrophosphorus are given.—R. A. R.

Pig Iron in Cupola Charges. L. J. White. (Institute of Australian Foundrymen : Australasian Engineer, 1947, Apr. 8, pp. 33–39). After explaining the blast-furnace process, in particular the manner in which the composition is affected by the practice employed, the author gives details of the standard and special irons made by the Broken Hill Proprietary Co., Ltd., and then describes the calculation of a cupola charge to produce iron to a specified analysis and the charging practice.—R. A. R.

Production and Properties of Nodular Cast Iron (Nodulite) : A Review of the Literature. G. R. Woodward. (Bulletin of the British Cast Iron Research Association, 1948, vol. 9, July, pp. 200–204). Abstracts are given of nine recent papers on the production and properties of nodular cast iron. J. C. R.

Cast Iron with Nodular Graphite. B. Tyberg. (Gjuteriet, 1948, vol. 38, Aug., pp. 114–119). [In Swedish]. A brief account is given of the manufacture and properties of cast iron with nodular graphite as reported by H. Morrogh and W. J. Williams (*see* Journ. I. and S.I., 1948, vol. 158, Mar., pp. 306–322).—R. A. R.

Modification of Cast Iron as a Means of Decreasing Rejected Castings. Ya. I. Preigerzon. (Vestnik Mashinostroeniya, 1947, vol. 27, No. 4, pp. 55–58 : Chemical Abstracts, 1948, vol. 42, July 10, cols. 4507–4508). Cast iron containing 5·5% (C + Si) for cylinder heads, pistons, and other engine parts was modified by the addition of 75% or 45% ferrosilicon. The addition of 0·4% of 75% ferrosilicon decreased the depth of chill from 7–12 to 4–7 mm, increased the tensile strength from 19·7 to 23·5 kg/sq.mm., and increased the Brinell hardness from 192 to 227. The percentage of castings rejected because of blowholes was also greatly reduced.

Tellurium-Graphite Use in Foundry Work. (Rly. Mech. Engr. 1948, vol. 122, pp. 83–84 : Railway Engineering Abstracts, 1948, vol. 3, Aug., p. 223). Reduction of car-wheel failures by over two-thirds in the number of wheels cast during ten years has been brought about in America by the addition of tellurium and graphite to the molten mixture in grey-iron foundry practice. The treatment is described and discussed ; it has been developed by the American Association of Manufacturers of Chilled Car Wheels.

Experiences in Operating the Speedmullor. (Foundry Trade Journal, 1948, vol. 85, July 15, pp. 59–60). A machine called the Speedmullor for mixing foundry sand, water, and binder is described. The mill is rubber lined and the two wheels in it are rubber tyred. It mixes batches of about 8 cwt at about 10 cu.ft./hr.—R. A. R.

Facing Sand Without Coal Dust. R. G. Bate. (Foundry Trade Journal, 1948, vol. 85, July 1, p. 6). To avoid dusty conditions in the foundry and obtain a superior surface finish to the castings, a synthetic sand was made using sand, clay, Fulbond No. 2 binder, cerial binder, and a little linseed-base core oil to prevent drying out. This gave satisfactory working properties and a good finish.—J. P. S.

HEAT-TREATMENT AND HEAT-TREATMENT FURNACES

Precision Heat Treating Performed at Rochester Division of Lindberg Steel Treating Co. (Industrial Heating, 1948, vol. 15, May, pp. 856–862 ; June, pp. 1039–1046, 1078). An illustrated description is given of the heat-treatment

equipment for ferrous and non-ferrous metals at the Rochester Works of the Lindberg Steel Treating Co., New York.—R. A. R.

Heating and Heat-Treating Operations in Production of Timken Railroad Bearings. (Industrial Heating, 1948, vol. 15, May, pp. 772–782). Some of the furnaces and heat-treating equipment used for making heavy roller bearings at the Timken Roller Bearing Company's works are described and illustrated.—R. A. R.

Heat Treating Research Aids Development and Production at Mack Manufacturing Corporation. C. C. Roberts. (Steel Processing, 1948, vol. 34, June, pp. 310–312, 319). Heat-treatment furnaces and temperature control instruments at the Mack Manufacturing Corp. are described. The heat-treatment units consist of a carburizing furnace, a brazing furnace, a tempering furnace, and two salt baths. All, except one salt bath, are electrically heated.—R. A. R.

Conveyor System Simplifies Coil Handling at Aliquippa. C. F. Seyler. (Iron Age, 1948, vol. 162, June 10, pp. 86–89). The conveyor system at the Aliquippa plant of Jones and Laughlin Steel Corp. is described. Designed to prevent edge damage, it incorporates automatic weighing devices and special units to facilitate scrap removal.—J. P. S.

Modern Heat Treating : A Reference Index. F. R. Morral. (Iron Age, 1948, vol. 162, June 10, pp. 83–85). This is a list, with index, of 105 articles which deal with : (*a*) The theory and mechanism of heat-treatment ; (*b*) the development and application of TTT curves ; (*c*) continuous cooling ; (*d*) the heat-treatment of cast and malleable irons ; (*e*) the Ms points of steel, determined and calculated ; (*f*) austenite transformation ; (*g*) martempering ; (*h*) isothermal heat-treatment in practice ; and (*i*) TTT curves for non-ferrous alloys, *e.g.* aluminium bronze.—J. P. S.

Recent Developments in Ferrous Metallurgy. J. W. Donaldson. (Journal of the West of Scotland Iron and Steel Institute, 1946–47, vol. 54, pp. 1–5). This paper briefly describes the more recent developments in the composition and heat-treatment of alloy steels, and the widening use of precision casting, induction heating, etc.—J. P. S.

Variation of the Rate of Cementation of Iron at the Curie Point. H. Forestier and G. Nury. (Comptes Rendus, 1948, vol. 227, July 26, pp. 280–282). In their examination of the variation of the rate of carburization of iron with temperature the authors used electrolytic-iron wires 0·25 mm in dia and 15-mm cubes of Armco iron polished to a mirror finish on one face ; carburization continued for 1 hr. with either town gas or carbon monoxide. Changes in the rate of reaction were measured by observing the electrical resistance of the wires and the hardness of the cubes. With the wires both gases produced a clear-cut maximum rate of reaction at the Curie point, which was particularly sharp with CO gas. The shapes of the Vickers hardness curves were conditioned somewhat by the depth of the indentation, but their indications agreed with those of the electrical resistance measurements.

Furnace Design for Better Utilization of Fuel. F. E. Harris. (American Gas Association : Metal Progress, 1948, vol. 53, June, pp. 817–822). **Needs in Furnace Development and Fuel Utilization for the Metal Industry.** F. E. Harris. (Steel Processing, 1948, vol. 34, June, pp. 313–317, 319). The theory of heat transfer is discussed and curves showing the increase in rate of heat transfer to steel with increase in furnace temperature are shown and the way in which this affects furnace design is pointed out. The gas-carburizing of clutch spring plates 8 in in dia and $\frac{1}{16}$ in thick, is described. R. A. R.

Heat Treating Furnaces. V. Paschkis. (Transactions of the American Foundrymen's Association, 1946, vol. 54, pp. 287–293). *See* Journ. I. and S.I., 1947, vol. 155, Jan., p. 150.

Salt Baths for the Treatment of Metals. T. A. Hood. (Australia, Department of Munitions, Munitions Supply Laboratories, 1947, Information Circular 9 : [Abstract] Bulletin of the British Non-Ferrous Metals Research Association, 1948, July, p. 220). The various salt mixtures, their typical applications, the design of furnaces, and materials of construction are described. There is a bibliography of 175 references, with abstracts of many papers.

Principles and Applications of Isothermal Heat Treatment. J. M. Hodge. (Society of Automotive Engineers : Steel Processing, 1948, vol. 34, July, pp. 375–376). Methods of modifying the isothermal heat-treatment of carbon and alloy steels so as to obtain the desired structure of fine or coarse pearlite, bainite, or tempered martensite are discussed. R. A. R.

Hardening with Atmospheres from Standard Raw Gas. J. G. Hoop. (Steel Processing, 1948, vol. 34, July, pp. 362–365). Some heat-treatment furnaces and controlled-atmosphere generators using natural gas as raw gas, are described. Of special interest is the "carbon potential" indicator which, by measuring the electrical resistance of a wire surrounded by the atmosphere, gives an indication of the carbon in solution.—R. A. R.

Through-Carburizing of Low Carbon Steel Permits Purchasing and Fabricating Economies. K. Rose. (Materials and Methods, 1948, vol. 27, June, pp. 68–71). A description is given of the methods of gas-carburizing developed by the General Motors Corp. for through-hardening low-carbon steel. An example is given of the treatment of clutch spring discs 8 in in dia, 0·057 in thick, of low alloy 0·15% carbon steel which were formed and then treated for 7 hr in an atmosphere of carbon monoxide, natural gas, and nitrogen, the composition of which was changed as the carburizing proceeded. The carbon content was raised to 0·45–0·50% right through the material.—R. A. R.

Carburizing and Nitriding Steels. R. F. Johnston. (Steel Processing, 1948, vol. 34, July, pp. 370–374). Suitable applications for case-hardened steels with initial carbon in the ranges 0·08–0·15%, 0·15–0·20%, and 0·20–0·30% are mentioned with notes on the physical properties. The particular merits of case-hardened alloy steels are dealt with next, and finally steels suitable for nitriding and equipment for nitriding, are described.—R. A. R.

Delayed Quench for Steel Castings. S. L. Gertsman. (American Foundrymen's Association, May 3–7, 1948 : Preprint No. 48-2). A "delayed quench" heat-treatment for cast steel ball sockets and the investigations necessary for its development are described. The steel contained approximately 0·40% of carbon and 0·85% of manganese. The Ac_3 and Ar_3 points, and the cooling rates in air of different parts of the casting were determined and the mechanical properties, grain-size, and microstructure at various stages of heat-treatment were studied. The treatment successfully applied to large numbers of these castings consisted of holding for $3\frac{1}{2}$ hr at 1700° F, cooling in air for 85 sec to 1350–1435° F, quenching in water and immediately tempering at 1225° F for $3\frac{1}{2}$ hr.—R. A. R.

Low Temperature Treatment of Steels. B. Hedde D'Entremont. (Machines et Métaux, 1948, vol. 32, July, pp. 221–226). The effect of cold-treating steels was studied by carrying out tests on tool steels (tungsten, tungsten–molybdenum, and tungsten–chromium–vanadium) and steels for dies and gauges (chromium the main alloying element).—R. F. F.

Flame-Hardening in the Manufacture of Gear Wheels. H. W. Grönegress. (Werkstatt und Betrieb, 1948, vol. 81, June, pp. 145–150). Illustrated descriptions are given of flame-hardening equipment for hardening gear teeth of different sizes. Data on the depth and degree of hardness and the consumption of gases are given and the advantages of the process are indicated.—R. A. R.

Flame Hardening—Principles, Applications, and Equipment. M. S. Rosengren. (Welding Journal, 1948, vol. 27, June, pp. 453–455). Principles and methods of flame hardening are described.—J. P. S.

Trends in Induction Heating. H. B. Osborn, jun. (Steel Processing, 1948, vol. 34, July, pp. 360–361). Under certain conditions induction heating has economic advantages ; the authors points out that old parts are being redesigned, and new parts designed, to facilitate the application of this method of heating for which he predicts an expanding future.—R. A. R.

Induction Hardening Expedites Production of Axle Shafts. G. O. Hunt. (Industrial Heating, 1948, vol. 15, May, pp. 758–762). A description is given of the induction-heating equipment of the Dana Corp., Fort Wayne, Indiana, for hardening automobile axle shafts. Three 125-kW machines have been installed, each with an output of 120 shafts/hr. The shafts are heated, by passing through the inductor block at 0·296 in/sec, to a depth of 0·250 in and fully hardened to a depth of 0·125 in.—R. A. R.

Mechanized Induction Heating. R. Vicker. (Steel Processing, 1948, vol. 34, June, pp. 302–303, 305). Some advantages of induction heating for heat-treating and other processes are

outlined with a brief description of the equipment used. R. A. R.

Skin Hardening by the High-Frequency Induction Process. R. J. Brown. (Iron and Coal Trades Review, 1948, vol. 157, July 9, pp. 57–62). The application of high-frequency induction heating to surface or skin hardening is described, and the formulae connecting penetration with frequency are given. Various types of oscillator are referred to, the preference in this country being for thermionic machines. The author also describes the structural changes taking place in steel during this form of heat-treatment.—J. P. S.

A New Method for the Surface Hardening of Steel by Induction Heating. M. G. Lozinsky. (Bulletin de l'Académie des Sciences, U.R.S.S., Classe des Sciences Techniques, 1946, No. 4, pp. 619–624).

High Frequency Furnaces for Heating Iron and Metals. Dreyfus. (Elektroteknikeren, 1946, vol. 42, July 27, pp. 337–220 : [Abstract] Centre de Documentation Sidérurgique, Bulletin Analytique, 1948, vol. 5, Mar., p. 10). A study is presented of the high-frequency furnace, its construction, the physical laws according to which it operates and the important qualities which enable it to compete with other furnaces.

Induction Heat Treatment Practice. F. P. Heard. (Australian Institute of Metals : Australasian Engineer, 1948, Mar. 8, pp. 43–50). Induction heat-treatment is comprehensively discussed from the following points of view : (1) Practical applications ; (2) surface hardening ; (3) depth of current penetration in iron ; (4) economy ; (5) equipment ; (6) power requirements ; (7) frequency ; (8) metallurgical aspects including, hardness ; microstructure ; and resistance to wear, shock, and fatigue ; (9) selection of steel ; and (10) steels for low-density heating.—R. A. R.

New Compound Hardens Steel. (Engineering and Mining Journal, 1948, vol. 149, July, p. 110). Reference is made to a new hardening powder called "Hi-Speed-It." Low-carbon steel is heated to cherry red and dipped in the powder for 15 to 30 sec. A film of powder fuses and clings to the steel which is again brought to red heat ; when the film spreads evenly over the steel it is quenched in cold water. It is claimed that nails treated in this way can be driven through $\frac{1}{8}$-in steel.—R. A. R.

The Properties of Oils Used in the Heat-Treatment of Steel. G. T. Dunkley. (Metal Treatment, 1948, vol. 54, Summer Issue, pp. 67–72). The author describes various properties of quenching oils in relation to their effect in heat removal and their fitness for continual use. Viscosity is among the most important, and should be as low as possible compatible with the absence of low-boiling-point constituents ; resistance to oxidation is also of considerable importance. Continuous cooling, combined with thermostatic control, maintains the best quenching properties and preserves the oil from deterioration.—J. P. S.

Quenching Steel in Molten Media. F. R. Morral. (Steel, 1948, vol. 122, June 21, pp. 92–95, 116). This paper reviews the literature on the quenching of steel in molten salt baths. There are 23 references.

"Submarine" Quench Technique Employed in Forming and Hardening Auto Leaf Springs. G. E. Stedman. (Steel, 1948, vol. 122, June 7, pp. 92–94). The United States Spring and Bumper Co., Vernon, California, uses a multi-station revolving unit called a "submarine" for quenching automobile spring leaves. The articles also describes the cold forming and plating of automobile bumpers.—J. P. S.

FORGING, STAMPING, DRAWING, AND PRESSING

Novel Billet Descaler Aids Forging. (Iron Age, 1948, vol. 162, July 15, p. 79). A simple machine is described for descaling small round billets before forging. A number of knives bear against the billet which is caused to rotate by friction against knurled rollers. There is little loss in heat during the process.—J. P. S.

Heavy Forging of Steel. (Iron and Coal Trades Review, 1948, vol. 156, June 18, pp. 1287–1290). The Chambre Syndicale de la Grosse Forge Française held a Symposium in Paris on May 27–29, 1948, entitled "Journées de la Grosse Forge." This report briefly reviews the following papers which were presented :

Improvement of the Soundness of Forging Billets by Reheating the Feeder Head, by Granger.

A New Method for Forging Crankshafts—The R.R. Process, by Vialle and Lafont.

Forging Manipulators, by F. J. Somers.

Fabrication of Large Parts by Mechanical Forging and Stamping, by Monnier.

Manufacturing Costs in Heavy Forging, by Friart.

Recent Tendencies in Large Forging Presses, by J. A. Sanderson.

Heat-Treatment of Large Forgings : Structure and Mechanical Properties as a Function of Speed of Heating, by Garçon.

Fabrication and Properties of a Forged Rotor in Special Steel, by H. H. Burton.

Refractory Linings of Reheating Furnaces, by Letort.

Modernization of a 6000-Ton Armour-Plate Press by Electro-Hydraulic Control, and Fabrication of High-Pressure Hollow Bodies, by Bonillot.

The Choice of Hammer. P. Patek. (Betrieb und Fertigung, 1948, vol. 2, Jan.–Feb., pp. 1–5). A method of calculating the number of blows required to produce a given forging from a blank of specified dimensions is explained and several examples of its use are given.—R. A. R.

Pipe Forming After Welding. G. E. Stedman. (Steel, 1948, vol. 122, May 24, pp. 98–100). Matched steel strips are fed above and below the flattened end of a brass mandrel, and the ends resistance-welded. The mandrel then tapers from a flattened oval to a circular section, in about 6 ft, and a tube is thus formed. Flanges protrude horizontally at the seams ; these are trimmed off, and the finished tube cut into required lengths with a flying saw ; it is then hot-dip galvanized.—M. A. V.

Press Fabricating Pipe. (Steel, 1948, vol. 123, July 26, pp. 84, 87). A press, 36 ft long, powered by six 500-ton jacks, forms $10\frac{3}{4}$-in pipe from $\frac{1}{4}$-in plate by bending it between a horizontal cylinder and a die of approximately 120° arc and the appropriate radius, in three stages. The edges are by then approximately $1\frac{1}{2}$–2 in apart, and a special welding mill with hourglass-shaped driving rolls brings them together and welds them by the submerged-arc process.—J. P. S.

Dutch Report on Steel Tube Making in Germany. (British Intelligence Objectives Sub-Committee, 1948. Miscellaneous Report No. 80 : H.M. Stationery Office). The report is presented by a team of Dutch investigators on the production of steel and strip for seamless and welded tubes, and on tube manufacture. Practice at the following works is described: A.G. Kronprinz Werke at Hilden ; Kronprinz A.G. für Metallindustrie at Solingen-Ohligs ; Stahlrohrenwerke Poppe und Pothoff ; Th. Kieserling und Albrecht Eisengiesserei und Maschinenfabrik at Solingen ; and Mannesmann Rohrenwerke at Witten and Hückingen.—M. A. V.

Factors in the Operation of a Mannesmann Piercing Mill. H. R. McLaren. (Iron and Steel Engineer, 1948, vol. 25, Apr., pp. 73–81). An outline is given of the piercing of billets for seamless tubes in the Mannesman process, and it is stressed that for smooth internal finish the plug must be set, in relation to the rolls, so that the steel does not break before it reaches it. The important points in the preparation, centring and heating of billets for the process, dressing of rolls, and design of piercer point are also described.—J. P. S.

Some Aspects of the Manufacture of Large Seamless Steel Tubes and Hollow Forgings. J. W. R. Naden. (Transactions of the Institute of Marine Engineers, 1948, vol. 60, July, pp. 151–162). The heating, hot-piercing and hot- and cold-drawing of large open- and closed-end cylinders, for pressure piping, boiler drums and steam receivers are described. J. P. S.

Spin 6-Inch Thick Steel Plate. (Machinist, 1948, vol. 92, July 10, pp. 303–304). The Lukens Steel Co. is operating a machine for producing elliptically dished tank heads up to 18 ft $2\frac{1}{2}$ in dia from plate. Heated discs of the appropriate diameter are pressed between a convex lower and a concave upper former which produce the crown, and the edge is formed by spinning between a movable outer roller and a fixed inner one. Illustrations show operations on material approximately $1\frac{1}{2}$ in thick.—J. P. S.

Hydraulic Stretch Forming Machine. (Sheet Metal Industries, 1948, vol. 25, Apr., pp. 766–767). Details and six illustrations of the Huffords hydraulic stretch-forming machine for the fabrication of components for jet aircraft

are presented. The overall length of the machine is 33 ft; each arm from tip to table is 14 ft long; the table is 4 ft 6 in wide, and the jaws will grip sheet up to 42 in wide. Each tension cylinder is of 50 tons capacity.—R. A. R.

Determining Time in Die-Forging. P. Patek. (Betrieb und Fertigung, 1948, vol. 3, Apr., pp. 61–65). A formula for determining the number of forgings per hour which can be made by die-forging is developed and examples of its application are given.—R. A. R.

Some Fundamental Considerations Relating to the Deep Drawing of Metals. A. R. E. Singer. (Sheet and Strip Metal Users' Technical Association : Sheet Metal Industries, 1948, vol. 25, July, pp. 1387–1393, 1400). The mechanism of deep drawing is examined by considering four cupping operations and the punch-load/punch-travel curves for each. These operations are: (1) Producing a cup from a blank of large diameter in a die with generous clearance so that no ironing of the wall takes place; (2) making a parallel-sided cup with some ironing of the wall; (3) reducing the thickness of the cup wall by causing an ironing action throughout the wall; and (4) producing a hollow body from a flat blank without a reduction in the periphery of the blank. The plastic deformation and mechanical properties of the metal and the lubrication necessary to meet deep-drawing requirements are discussed in detail. Finally the application of the results of fundamental research to problems of press-shop production is dealt with.—R. A. R.

Strain Measurements in Deep Drawing. K. L. Jackson. (Sheet Metal Industries, 1948, vol. 25, Apr., pp. 723–724). A method of assessing blank profiles of almost any shape by direct measurement of the plastic strain during drawing is described. This is possible by fixing a photogrid on to the unformed blank. About 1·5 c.c. of a photographic emulsion are poured on to the centre of the cleaned blank which is then spun at 400–500 r.p.m. to spread it. After drying, a glass negative marked with a grid is pressed on the emulsion with a heavy glass block and the system placed below a 500-W mercury-vapour lamp. The exposure is 6 min. The specimen is washed and placed in a solution of methyl-violet in water for 1 min. After rinsing it is ready for use. The change in the distance between the grid line after drawing or pressing gives the plastic strain.—R. A. R.

The Routine Checking of Crank Presses. K. L. Jackson. (Sheet Metal Industries, 1948, vol. 25, July, pp. 1353–1354). Recommendations are made on the maintenance and repair of crank presses.—R. A. R.

Induction Heating for Steel Forging Can Reduce Costs. K. Rose. (Materials and Methods, 1948, vol. 27, Apr., pp. 76–80). The increased cost of induction heating in forge furnaces, as compared with other types of heating, may be offset by the resultant advantages, particularly the virtual elimination of scaling, which is especially important when alloy steels are being forged, and the greater possibilities of automatic operation and control. The disadvantages, apart from cost, are the necessity of forging the steel in one operation, and the difficulty of aligning coils for irregular forms. The process is best applied to long runs, and not to a job-shop schedule with short runs.—M. A. V.

Elimination of Shatter Cracks by Hot Working. E. L. Koehler and H. B. Wishart. (Transactions of the American Society for Metals, 1948, vol. 40, pp. 513–528). Rolled ingots of chromium–molybdenum steel (C 0·41%, Cr 0·94%, Mo 0·19%) in some of which shatter cracks formed during cooling were tested to determine whether the cracks would weld together on subsequent hot forging or rolling. These cracks could be welded up by hot work in either form but the reduction required to produce the weld was less with forging than with rerolling. If the shatter cracks are unfavourably orientated they may be spread open by the hot working. R. A. R.

Investigations of German Heavy Forging Plants. (British Intelligence Objectives Sub-Committee, 1948, Final Report No. 1398 : H.M. Stationery Office). This report covers the machinery and processes at German works producing heavy forgings. It also includes descriptions of the casting of composite ingots, centrifugal casting, the "Torsator" torsion-pulsator fatigue-testing machine capable of applying twisting moments up to 2000 mkg, and heavy lathes, boring, and planing machines.—R. A. R.

Cold Heading Cuts Manufacturing Costs. C. S. Ricker. (Machinist, 1948, vol. 92, June 12, pp. 195–197). Some advantages of cold heading in the manufacture of bolts and studs of various shapes are enumerated with data on the material and time saved as compared with machining operations. R. A. R.

Wire-Drawing. H. Richards. (Journal of the Institution of Production Engineers, 1948, vol. 27, July, pp. 333–351). A comprehensive account is given of the processes and equipment for drawing ferrous and non-ferrous metal wires.—R. A. R.

Heating Phenomena in Wire Drawing. H. Korst. (Betrieb und Fertigung, 1948, vol. 2, July, pp. 106–108). The formula developed by Siebel and Kobitzsch for determining the increase in the temperature of wire as it is drawn through a die is discussed with examples of its use.—R. A. R.

A New Theory of the Plastic Deformation in Wire-Drawing. R. Hill and S. J. Tupper. (Journal of The Iron and Steel Institute, 1948, vol. 159, Aug., pp. 353–359).

Plastic Deformation of Carbon Steels. R. Saxton. (Metallurgia, 1948, vol. 37, Apr., pp. 293–294). The preparation of the material and the subsequent cold-working of carbon steel by wire-drawing are shortly described. Changes in the crystal structure brought about by the wire-drawing are noted. The torsion test (used in England) and the flexion test (used on the Continent) for examining the finished product, are touched on.—A. E. C.

Wire Drawing Problems—Drawing Solutions and Die Life. (Industrial Diamond Review, 1948, vol. 8, Apr., pp. 101–102). Experience with a proprietary lubricant and a soluble oil, both mixed with water, in drawing wire through carbide dies is reported. The precise nature of the drawing compounds is not stated.—M. A. V.

Wire Mill Practice. W. F. G. Kerley. (Wire Industry, 1947, vol. 14, Aug., p. 443; Dec., pp. 683–684; 1948, vol. 15, July, pp. 457–458). A simple method for the calculation of reductions in wire-drawing dies is explained with practical examples in Parts I and II, and Part III is devoted to the atomic and crystalline structures of metals and their relation to die wear.—R. A. R.

New "Boldalloy" Wire Spooling Machine. (Wire Industry, 1948, vol. 15, June, p. 391). This machine is designed to wind wire and tape on to spools evenly and compactly; ingenious features of mechanical design are said to overcome the defects of earlier machines.—J. P. S.

"Kneller" Automatic Die Polishing Machine. (Wire Industry, 1948, vol. 15, June, p. 379). This article describes a new machine for polishing diamond or tungsten-carbide dies up to $\frac{1}{4}$ in dia using either solid wires, or flexible abrasive carriers for the purpose. A true lapping action is maintained automatically.—J. P. S.

The Administration of Inspection and Despatch in a Steel Wire Mill. C. Coates. (Wire and Wire Products, 1948, vol. 23, Apr., pp. 307–310, 346–348).

Wire and Processing Equipment in the 1948 B.I.F., Birmingham. (Wire Industry, 1948, vol. 15, May, pp. 317–319). Some of the equipment exhibited is discussed. This includes a Marshall Richards 9 MS2 wire-drawing machine with DS 120D spooler, a nine-die slip machine with the drawing cones and dies completely submerged; a 9 FS2 machine with PS4 spooler; a new totally immersed plating barrel (W. Canning and Co., Ltd.), and a gas-fired bolt-heating machine.—M. A. V.

ROLLING-MILL PRACTICE

The Rolling of Metals : Theory and Experiment—Part XV Discussion of Certain Practical Rolling Problems in the Light of the Theory of Rolling. Part XVI A Survey of Present Knowledge and the Direction of Future Research. L. R. Underwood. (Sheet Metal Industries, 1948, vol. 25, Feb., pp. 297–302, 308; Mar., pp. 497–502; Apr., pp. 704–706, 716; May, pp. 917–921). Conclusion of a series of articles (*see* Journ. I. and S.I., 1948, vol. 158, Mar., p. 400).

Calculation of the Forces and Power Involved in Rolling on the Basis of Test Results. A. Geleji. (Schweizer Archiv, 1947, vol. 13, Nov., pp. 336–344). Formulæ are developed for calculating the power requirements, roll pressure, and deformation resistance when rolling square bars with cylindrical rolls and no side pressure. Their practical application is demonstrated using data from rolling tests on different metals, and good agreement between theory and practice is found. (*Note*: An abridged translation appeared in Engineers' Digest (London), 1948, vol. 9, Apr., pp. 126–129).

Application of the Basic Principles of Rolling in Roll Design. R. E. Beynon. (Iron and Steel Engineer, 1948, vol. 25, June, pp. 37–59). After discussing the basic principles of rolling, especially the relation between roll diameter and the elongation and spread of the material, the author discusses the rolling of sections, such as beams, channels and angles, tees and rails, and the principles to be followed in designing roll passes for these purposes.—J. P. S.

The Rôle of Forward Slip and Lag during Rolling. A. Groza and Z. Wusatowski. (Hutnik, 1947, vol. 14, Nov., pp. 512–518). [In Polish]. This article discusses the plastic deformation of metal during rolling, and explains methods of calculating the angles of contact of the bar with the rolls and the position of the neutral axis. The increased plasticity at high temperatures and the lowered friction cause forward slip at the surface, leading to splitting and lamination in the interior. W. J. W.

Operation and Maintenance of Variable Voltage Control Systems. A. D. Howry. (Iron and Steel Engineer, 1948, vol. 25, May, pp. 48–55). This paper describes the operation and maintenance of: (*a*) A blooming-mill drive with variable voltage control; and (*b*) screw-down operation with rotating regulator variable voltage control.—R. A. R.

Large Electronic Direct Current Motor Drives. M. M. Morack. (Iron and Steel Engineer, 1948, vol. 25, June, pp. 80–85). Rectifying devices, based on the use of sealed ignitions have been developed which permit the use of adjustable-voltage electronic direct-current motor drives where reversal is not required.—J. P. S.

The Speed of Electric Drives for Coilers on Cold Rolling Mills. A. N. Troshnikov. (Vestnik Electropromyshlennosti, 1947, No. 8, pp. 8–9).

Design, Construction and Lubrication of Mill Couplings and Spindles. W. L. Stover. (Iron and Steel Engineer, 1948, vol. 25, July, pp. 62–68). Recent designs of spindles and couplings for rolling-mill drives are described and illustrated. R. A. R.

Induction Motors for Rolling Mills. F. Landau. (A.S.E.A. Journal, 1947, vol. 20, Nov.–Dec., pp. 159–178: Electrical Engineering Abstracts, 1948, vol. 51, Apr., p. 93). This is a survey of the electrical systems and mechanical equipment used in reversible and non-reversible rolling mills with descriptions of gear units, pinion stands, shaft coupling, etc. An analysis is given of the torque distribution between flywheel and motor. Slip regulation and braking are considered. Photographs and diagrams of the plant and systems are shown.

Operation and Maintenance of Automatic Preset Plate Mill Screwdown Control; Electrical Features. J. F. Skalka. (Iron and Steel Engineer, 1948, vol. 25, July, pp. 37–40). The electrical control equipment for presetting the screwdowns of the 120-in plate of the Lukens Steel Co. is described and illustrated. Presetting means that a sequence of roll openings can be set up for a series of passes, so that by merely pressing a button, once after each pass, the roll opening is automatically adjusted to the next pass.—R. A. R.

Operation and Maintenance of Automatic Preset Plate Mill Screwdown Control; Mechanical Features. R. G. Uhler. (Iron and Steel Engineer, 1948, vol. 25, July, pp. 40–42). Drawings of the screwdown mechanism of a 120-in plate (*see* preceding abstract) are presented and some difficulties encountered with regard to lubrication are discussed.—R. A. R.

Structural and Rail Mill Rolls—Uses and Applications. A. F. Eisel. (United Effort, 1948, vol. 28, June, pp. 6–8). Types of material for rolls for various purposes are described briefly; it is important to consider all the dimensions and the drafts to be taken, in choosing the material.—J. P. S.

Modern Universal Slabbing Mills. A. R. Kruse. (United Effort, 1948, vol. 28, May, pp. 3–5). A description is given of a modern universal slabbing mill designed to roll ingots up to 36 in thick × 66 in wide × 108 in long either on edge or flat. The horizontal rolls are 45 in in dia × 80 in long, and the vertical rolls 36 in in dia × 80 in long. The opening between the latter can be varied between 20 and 80 in. The former rolls run in roller bearings and the latter in bronze bearings.—R. A. R.

New 28-in Reversing Mill for Alloy Steels. T. Bishop. (Iron and Coal Trades Review, 1948, vol. 157, Aug. 6, pp. 287–292). An illustrated description is given of a 28 in by 84 in hot reversing blooming mill designed for the production of alloy steels slabs and billets at the works of Thos. Firth and John Brown Ltd. The components of the mill train, the mill spindles, roll housings, manipulators, the lubrication system, and the sliding-type horizontal hot saw are all described in detail.—J. C. R.

Sheet and Tinplate Mills—Measurement of Roll Camber. J. H. Mort. (Iron and Steel, 1948, vol. 21, Apr., pp. 111–115; May, pp. 147–149). This paper describes the methods followed in grinding rolls, both to convex and concave cambers, and the derivation of useful tables for the purpose. J. P. S.

Sheet Leveler Incorporates New Feature. T. E. Lloyd. (Iron Age, 1948, vol. 161, May 20, pp. 92–94). The Voss-United chain-type roller leveller incorporates solid alloy cast-iron platens for backing up in place of the more usual back-up rolls, and an endless double-sprocket chain-roll drive. The lower roll assembly can be tilted bodily to facilitate admission of the sheet. The leveller described is designed to flatten mild-steel sheets of yield point 35,000 lb/sq.in., 0·025–0·075 in thick, and up to 28 in wide, at a speed of 75–300 ft/min.—M. A. V.

Driven Back Up Rolls Used in Cold Strip Mill. (Iron Age, 1948, vol. 162, July 15, pp. 90–91). Driven backing rolls and front and back reel tension overcome some of the disadvantages of the Steckel mill. A 10-in mill having 3-in work rolls and 14-in backing rolls is described; its production on light gauges is 3 to 10 times that of a similar Steckel mill.—J. P. S.

Cone Worm Reducers and Gear Sets Avoid Breakage and Mill Shutdowns. E. C. Denne. (United Effort, 1948, vol. 28, June, pp. 3–4). A type of worm gear is described in which the teeth have straight sides, tangential to a common circle; a greater number of teeth are engaged at all times and thus torque transmitted may be increased, and wear is reduced. J. P. S.

Rototrol in Steelmaking. (Westinghouse Engineer, 1948, vol. 8, May, pp. 73–74: [Abstract] Metals Review, 1948, vol. 21, June, p. 32). Used for the control of a variety of steel-mill equipment, it is essentially a small D.C. generator. Its application to blooming mills and pickling lines is described.

Specifying and Standardizing Rolled Steel Products. P. M. Waszink. (Metalen, 1948, vol. 2, May, pp. 196–202). [In Dutch].

MACHINERY FOR IRON AND STEEL PLANT

A Thrustor Lowering Control System for A.C. Crane Hoists. E. L. Schwarz-Kast. (Transactions of the American Institute of Electrical Engineers, 1947, vol. 66, pp. 139–144). Illustrated particulars are given of the Thrustor, an electrohydraulic device developed by the General Electric Company for actuating servo-mechanisms of any kind, and especially for brake release.—R. E.

Journal- and Roller-Bearings and Couplings for Electric Drives. K. Bobek. (Elektrotechnik und Maschinenbau, 1944, vol. 62, Feb. 18, pp. 83–91: Electrical Engineering Abstracts, 1948, vol. 51, Aug., p. 220). Fundamentals of journal- and roller-bearing operation and clearances are discussed. Journal bearings permit shafts to take up different positions when running and starting. This must be taken into account when choosing suitable couplings. Various right and wrong coupling applications are given, and a common mistake with motor-driven boiler feed pumps is exposed. For oil-cooling of bearings the pressure should be $\geq$ 0·3–0·5 atm to prevent spillage. Pressure lubrication is used frequently on starting or reversing large machines. Types and reasons for choice of bearing are given, and electrical and mechanical causes and typical appearances of bearing damage are listed.

Operation of Ingot Buggies by Remote Control. K. S. Kuka. (Iron and Steel Engineer, 1948, vol. 25, June, pp. 77–79). A system is described, in use at the Tata Iron and Steel Company, at Jamshedpur in India, by which ingot cars are controlled, on their way from mill to furnaces, in such a way that the furnace can be selected in advance and the car accelerated, decelerated and stopped as desired. Safety devices are installed to eliminate danger from faults to earth and short circuits.—J. P. S.

New "Barco" Straightening and Cutting Machine. (Wire Industry, 1948, vol. 15, June, pp. 389–390). A new machine is described for straightening wire of 0000 to 10 S.W.G. and

cutting it into lengths of 6½ in to 19 ft as required. Twenty-four possible combinations of rate of wire feed with spinner speed are made by use of a two-speed main drive, a three-speed gear change and multi-link V-belts on the spinner shaft. J. P. S.

Shading Coil Calculations for Single-Phase Magnets. C. T. Evans. (Transactions of the American Institute of Electrical Engineers, 1947, vol. 66, pp. 134–138). A procedure is suggested by means of which the watt loss in the shading coil of a single-phase magnet may be calculated with reasonable accuracy. The validity of these approximate calculations is confirmed by experiments which are described.—R. E.

WELDING AND FLAME-CUTTING

Welding Metallurgy. R. J. Gillespie. (Canadian Machinery, 1947, vol. 58, Aug., pp. 134, 136 : [Abstract] Centre National de la Recherche Scientifique, Bulletin Analytique, 1948, vol. 9, p. 1295). Metallurgical principles for welding carbon steels, low-carbon alloy steels, cast steels, cast irons, and tool steels are outlived.

The Brittle Transition Temperatures of Various Low Carbon Steels Welded by the Same Method. N. Grossman and C. W. MacGregor. (Welding Journal, 1948, vol. 27, May, pp. 267-s–271-s). From tests made by welding, by the Unionmelt process, plates of seven low-carbon steels and by submitting them to notch bend tests with notches at varying distances from the weld, and at various temperatures, it is found that, in all cases, the weld metal is more ductile than the base plate, its transition temperature varying from −95° to −180° F (−70° to −118° C) and that of the base metal from −50° to −68° F (−45° to −55° C) in the least affected part ; at the most affected part, 1 in from the centre line of the weld, it was from −30° to −55° F (−34° to −48° C). Variations in the chemical composition of the deposited weld metals seems to be responsible for the variation in its transition temperature which can be correlated with both the manganese/carbon ratio and the nitrogen content. No such correlation exists for the steels ; it can be said, however, that the semi-killed steel is worse than the rimmed steel at the most brittle region though similar to it in the unwelded state. J. P. S.

Factors Affecting the Weldability of Carbon and Alloy Steel. Part I—Development of Test Procedure and Effect of Composition. C. M. Offenhauer and K. H. Koopman. (Welding Journal, 1948, vol. 27, May, pp. 235-s–252-s). Experiments with slow notch bend tests have shown that the factors influencing the tendency to a brittle fracture come within the following four groups : (*a*) The stress system or the degree of restraint imposed in the construction ; (*b*) the temperature of service ; (*c*) the rate of loading ; and (*d*) the strength and hardness. The transition temperature rises with increased carbon content, grain-size, hardness, rate of loading, and degree of restraint. In bend tests, where all variables save temperature are held constant, the embrittling temperature can be defined as that at which a lateral contraction of 1% occurs ; at this point the ratio proportional-limit/maximum-strength approaches unity.—J. P. S.

Factors Affecting the Weldability of Carbon and Alloy Steels. Part II—Effect of Welding Technique on the Transition Behaviour of Welded Structures. C. E. Jackson and W. J. Goodwin. (Welding Journal, 1948, vol. 27, May, pp. 253-s–266-s). Tests have been made by laying down beads of weld metal on a number of steels with a range of speeds of traverse, current density, and voltage. It is concluded that, for a given energy input per unit length, high speeds and high currents produce a bead of larger section than low speeds and low currents ; a lower hardness will be obtained in the heat-affected zone, as this is related to the area of the bead and the transformational characteristics of the steel. The temperature for transition from ductile to brittle fracture is raised by an increase in the maximum hardness of the heat-affected zone, an increase in its grain-size and by an increase in the width of the heat-affected zone through which the failure-crack is propagated. These factors may be independent ; thus, a decrease in the width of the heat-affected zone may compensate for an increase in its hardness so that welds produced by different techniques may have the same transition temperature.—J. P. S.

The Effect of Welding and Other Local Heating Processes on Residual Stress and Dimensional Changes in Steel. L. C. Bibber. (Steel Processing, 1948, vol. 34, May, pp. 251–255, 268). Methods of avoiding, and reducing, locked-up stresses in welded structures are discussed.—J. P. S.

A Summary of the Fuller Carbo-Flux Process for Welding Light Gauge Stainless Steel. (South African Welder, 1948, May, pp. 17–21). A process approved by the Air Ministry for the welding of stainless sheet not thicker than 14 S.W.G. is described. It is known as the Fuller Carbo-Flux process. A carbon electrode and D.C. are used, and a flux mixed with water or alcohol is painted on to the edges to be welded. R. A. R.

The Production of Seamwelded Pipes. R. Glass. (South African Welder, 1948, May, pp. 14–15). A brief description is given of the continuous process for the forming and oxy-acetylene welding of steel strip into tubes up to 3-in dia. R. A. R.

Submerged Melt Welding Applied to Hardenable Steels. E. A. Clapp and E. L. Frost. (Canadian Metals and Metallurgical Industries, 1948, vol. 11, May, pp. 19–21). The advantages of the submerged-melt electric welding process, in which a continuous stream of a granulated flux is fed over the end of the electrode, are outlined, and it is shown that the flexibility of control enables many hardenable steels to be welded without preheating.—R. A. R.

Manual Hidden-Arc Welding. J. L. Wolcott. (Steel 1948, vol. 122, May 10, pp. 102, 112). Apparatus for the manual use of hidden-arc welding has been developed by the Lincoln Electric Co., Cleveland, Ohio ; a conical hopper about 1 ft long by 5 in max dia contains the granular flux, which feeds by gravity ; it is equipped with a handle for manipulation. The electrode, $\frac{5}{64}$-in wire, is fed at 300 ft/min through this handle to the tip of the cone. Constant voltage control enables the operator to raise or lower the implement, thus controlling the flow of flux.—J. P. S.

Some Aspects of Welding in the Heavy Chemical Industry. E. Fuchs and D. A. Godfrey. (Transactions of the Institute of Welding, 1948, vol. 11, May, pp. 97–107). A number of applications of welding in the construction of tanks, bunkers, heat exchangers, driers, etc., in the chemical industry, and the building-up and repairing of worn shafts, castings and so on, are described.—J. P. S.

High Manganese Steel and Its Deposition by Arc Welding. R. W. Edwards. (Metallurgia, 1948, vol. 38, May, pp. 12–14, 57). The characteristics of high-manganese steel and its deposition by arc welding are discussed. Using electrodes containing Mn 11–15%, C 0·8–1·4%, and Ni 3–5%, metal may be deposited which does not require quenching, work-hardens adequately, and resists abrasion and high temperatures.—J. P. S.

A Method of Metal Welding with Separate Melting Processes. V. P. Nikitin. (Comptes Rendus (Doklady) de l'Académie des Sciences de l'U.R.S.S., 1947, vol. 56, No. 5, pp. 477–480 : Railway Engineering Abstracts, 1948, vol. 3, Aug., p. 213). A novel method is proposed, in which a concentrated heat source preheats and softens to a given depth the surface of the base metal, while a liquid jet of welding metal is fed into the seam from a special fixture. Details of the automatic installation are described, and schematic and sectional drawings are given of the main welder with a magnetically rotated arc and of the " dosing " fixture which squirts liquid metal through tubes having a diameter of at least 1·4 mm.

Three Phase Balanced Load Resistance Welding. C. A. Burton. (Welding, 1948, vol. 16, May, pp. 184–192). This paper describes circuits for the application of three-phase current to welding transformers and the apparatus constructed for the purpose by Sciaky Electric Welding Machines, Ltd., of Slough.—J. P. S.

Voltage Changes Caused by Resistance Welding Loads. I. B. Johnson, H. A. Peterson, and C. M. Rhoades, jun. (Transactions of the American Institute of Electrical Engineers, 1947, vol. 66, pp. 664–676). The authors present data, largely in the form of curves, relating to the effects of various welding loads in producing voltage changes.—R. E.

Functionalized Resistance Welding Control. C. B. Stadum, W. E. Large, and E. C. Hartwig. (Transactions of the American Institute of Electrical Engineers, 1947, vol. 66, pp. 414–418). By using an alternating voltage as a source of supply for timing circuits it was found possible to eliminate the prob-

lems of supply-voltage stability which are present in controls using a D.C. supply.—R. E.

Electrical Characteristics of Resistance Welding Tools. E. M. Callender. (Transactions of the American Institute of Electrical Engineers, 1947, vol. 66, pp. 393–404). Methods of electrical measurement and induction calculation of resistance welding tools are described.—R. E.

The Voltage Drop in the Welder Throat Loaded with Ferromagnetic Materials. G. H. Stein. (Transactions of the American Institute of Electrical Engineers, 1947, vol. 66, pp. 356–362). The resistance welding machine can be considered as a single-phase transmission system. Its power is supplied through the welding transformer and the welder throat represents the transmission line; the welding spot can be considered as the load. Each of these three elements will consume a resistive and reactive voltage component whereby the overall efficiency is relatively low and the regulation becomes very high as compared with a regular power-transmission system. The author has calculated the influence of the welding materials on the fields and voltages of the welder throat, and compared the computed values with the test results of J. J. Riley and C. E. Smith. (*Ibid.*, 1946, vol. 65, pp. 852–860).—R. E.

Hard Surfacing with High Frequency Currents. E. M. Kouzmak and A. I. Kourdin. (Avtogennoe Delo, 1947, No. 3, pp. 1–4: Engineers' Digest (London), 1948, vol. 9, Aug., pp. 258–260). Brief details are given of a method of applying high-frequency current to weld a layer of hard-metal alloy, formed from tungsten-carbide powder and borax, on to steel. The heating coil used for facing oil-well drills is described. The hardness and life of the coating were better than when the same alloy was deposited by gas or arc welding.
R. A. R.

Recommended Practices for Salvaging Automotive Gray Iron Castings by Welding. (Welding Journal, 1948, vol. 27, May, pp. 351–358). This article describes the use of cast iron, nickel, and Monel metal electrodes for the repair of defective castings, and also refers to hard soldering, metal spraying and to mechanical methods such as plugging and dowelling.—J. P. S.

Bronze-Welding of Cast Iron. E. Ryalls. (Welding, 1948, vol. 16, May, pp. 203–210). Details are given of various techniques for bronze welding cast iron with the oxy-acetylene torch and filler rods of copper zinc, manganese bronze, and nickel bronze. The need for care in joint preparation, preheating, and tinning is emphasized.—J. P. S.

Heat Processing Easily Oxidized Metals. F. C. Kelley. (Iron Age, 1948, vol. 161, May 20, pp. 84–89). Difficulties are encountered in furnace-brazing alloys such as high chromium steels and 18/8 stainless steel, owing to attack by oxygen, water-vapour, and nitrogen. These difficulties can be overcome by brazing in a stream of pure dry hydrogen, which is capable of reducing Cr_2O_3, without the use of a flux, any water vapour formed being drawn off. A number of typical furnace accessories are described and illustrated.
M. A. V.

The Radiography of Welds. R. F. Bishop. (Transactions of the Institute of Marine Engineers, 1948, vol. 60, June, pp. 131–142). Modern types of X-ray tube are described, and methods of examining welds in various kinds of structure, *e.g.*, castings, boiler drums, and large and small diameter pipe.—J. P. S.

Belgian Tentative Specifications for Welded Structures. (Revue de la Soudure/Lastijdschrift, 1947, vol. 3, No. 4, pp. 188–195; 1948 vol. 4, No. 1, pp. 39–50; No. 2, pp. 82–106). Tentative rules issued by the Institut Belge de Normalisation for the preparation of materials, methods of welding, and methods of testing are outlined.—R. F. F.

CLEANING AND PICKLING

Continuous Strip Pickling. E. D. Martin. (American Iron and Steel Institute, May 26–27, 1948, Preprint). Details of the development of the continuous pickling plant for mild carbon steel strip at the Inland Steel Co. are given. The effect of various factors on the sulphuric acid pickling process was investigated. The rate of pickling hot samples from the mill was much higher in solutions of high acidity (above 16% by volume). At certain ranges of acidity and temperature the pickling rate was practically independent of the iron sulphate content. Maximum rates were obtained at 190° F. with about 21% by volume of H_2SO_4, and at 260° F at about 18% by volume. The higher the temperature, the greater is the pickling rate in the 180–220° F. range. A theory to explain the effect of iron sulphate content on the pickling activity is offered. Finally an investigation of acid regeneration systems is reported and the regeneration plant designed on the basis of the results is described.—R. A. R.

A Modern Continuous Pickling Plant. R. Gille. (Electricité, 1948, vol. 32, Jan., pp. 1–2).

Fabricating and Finishing Stainless Steel. A. P. Schulze. (Metal Finishing, 1948, vol. 46, Jan., pp. 72–77; Feb., pp. 59–64; Mar., pp. 62–75; May, pp. 64–68; June, pp. 103–109). The properties of stainless steels are discussed and descriptions are given of methods of cleaning them with emulsions and alkalis, by electrolysis, sand-blasting, pickling, grinding, and buffing. A method of blackening stainless steel is described with data on the corrosion resistance of the treated surface in weak nitric and hydrochloric acids. Finally, the soft soldering, silver soldering, welding, and etching of stainless steel are briefly dealt with.—R. A. R.

Electrolytic Pickling. K. Weller. (Metal Finishing, 1948, vol. 46, June, pp. 95–102). The advantages and disadvantages of chemical and electrolytic pickling are compared and a large number of electrolytic processes (anodic, cathodic, and with alternating current) are reviewed.—R. A. R.

Industrial Hazards and Sickness in Iron and Metal Pickling Shops. Hermine Vogel. (Metalloberfläche, 1948, vol. 2, Apr., pp. 87–90).

Flame Cleaning: A Modern Method of Preparing Steel for Painting. G. W. Rigby. (Coke Oven Managers' Association: Gas World, 1948, vol. 128, June 5, pp. 7-s–18-s). It is suggested that flame-cleaning is the best method of removing moisture and mill scale from steel prior to painting. The process is described, and data on labour and consumption given.
M. A. V.

PROTECTIVE COATINGS

Electroplated Coatings. G. Black. (Materials and Methods, 1948, vol. 27, June, pp. 93–104). These notes and comprehensive tables summarize available information on: (1) Properties of coatings; (2) types and characteristics of electrolytes; (3) electroplating theory; (4) preparation for electroplating; (5) equipment for electroplating; and (6) testing of coatings.
R. A. R.

Determining the Weight and Thickness of Metallic Protective Coatings by Chemical and Electrolytic Dissolution. (Metalloberfläche, 1948, vol. 2, May–June, pp. 120–122). A list is given of solutions and their compositions for the removal of metallic coatings from steel and other metals so that their weight can be determined by difference. Methods of ascertaining their thickness (*a*) by determining the amount of hydrogen evolved in dissolving the coating from a given area, (*b*) from the time required for removal by electrolysis, and (*c*) chemical drop tests, are also described.—R. A. R.

Magnetic Measurement of the Thickness of Composite and Nickel Coatings on Steel. A. Brenner and E. Kellogg. (Journal of Research of the National Bureau of Standards, 1948, vol. 40, Apr., pp. 295–299). *See* Journ. I. and S.I., 1948, vol. 159, June, p. 223.

The Adhesive Strength of Electrolytic Deposits. R. Erdmann. (Metalloberfläche, 1948, vol. 2, May–June, pp. 117–119). Factors affecting the adherence of electrolytic coatings of zinc, copper, tin, and aluminium on steel and other metals are discussed and pretreatments which improve the adherence are described.—R. A. R.

Measurement of Adherence of Organic Coatings to Metal Surfaces. H. Green and T. P. Lamattina. (Analytical Chemistry, 1948, vol. 20, June, pp. 523–527). An adherence tester is described in which a metal plate covered with the coating under test, is drawn along against a knife edge, 4 mm wide and perpendicular to the coating. The reaction on the knife edge is measured by means of the distance a weight is raised. The force due to friction being subtracted, the result is expressed in dynes per cm width. Correction for the force required to tear the coating at the end of the strip can be calculated and applied but is insignificant where film thickness is small.—J. P. S.

Automatically Controlled Copper-Oxide Rectifiers for Electroplating and Anodizing Applications. J. J. Buckley.

(Transactions of the American Institute of Electrical Engineers, 1947, vol. 66, pp. 181–184). An illustrated description is given of the copper-oxide rectifiers used in large electroplating and anodizing installations. Several copper-oxide rectifier units and an induction-type control are dealt with in detail, and a brief résumé is given of rectifier fundamentals. R. E.

Drying with Infra-Red Rays. (Österreichischer Maschinenmarkt und Eisenhandel, 1948, vol. 3, June, pp. 161–166). This is a general review of the theory and practice of infra-red drying with descriptions of the equipment used.—R. A. R.

Drying Glossy Coatings with Infra-Red Rays. H. Weise. (Metalloberfläche, 1948, vol. 2, May–June, pp. 112–113).

The Industrial Applications of Hard Chromium. P. Morisset. (Machines et Métaux, 1948, vol. 32, Aug., pp. 281–286). The properties of hard chromium coatings and their industrial applications are surveyed.—R. F. F.

Diesel Engine Wear Speeded by Surface Disintegration Slowed by Porous Chrome Plating. W. G. Payne and W. F. Joachim. (S.A.E. Journal, 1948, vol. 56, July, pp. 40–43). The causes of wear on Diesel-engine cylinder liners are given and the improvements obtained by chromium plating and etching the lining are pointed out.—R. A. R.

Coating Steel with Nickel by Immersion in Nickel Chloride Solutions. W. A. Wesley and H. R. Copson. (Journal of the Electrochemical Society, 1948, vol. 94, July, pp. 20–31). A simple method for depositing controlled amounts of nickel on steel from a hot nickel-chloride boric-acid solution is described. Deposits up to 0·7μ in thickness can be readily produced; simple heat-treatment renders them compact and adherent. Steel coated in this way rusts readily, but its surface is altered sufficiently to suggest that it makes a better basis for organic finishes, conversion coatings, and ceramic enamels.—R. A. R.

Tinning of Malleable Iron. (Fonderie, 1946, July, pp. 267–268). Whiteheart and blackheart malleable iron having a decarburized surface can be tinned without difficulty. With ordinary blackheart malleable iron, the surface must first be cleaned by pickling and polishing and then covered with a thin metal layer, preferably by electrolysis. The castings are next dipped in boiling zinc chloride or ammonium chloride solution, and finally immersed in liquid tin.—J. C. R.

New Developments in Tin Plate Manufacture. E. F. Harris. (American Society of Mechanical Engineers: Steel, 1948, vol. 121, Apr. 12, pp. 94–96). Descriptions are given of mechanical equipment for electrolytic pickling, sorting, counting, and weighing sheets of tinplate at a large hot-dip tinning plant.—R. A. R.

Solder Stops Steel Tube-Sheet Corrosion. E. R. Stauffacher. (Electrical World, 1947, vol. 128, Aug. 16, p. 66: [Abstract] Corrosion, 1948, vol. 4, June, p. 19). Of the several protective coatings tested to prevent corrosion on mild-steel tube sheets used with aluminium–brass condenser tubes in sea water, 50–50 solder plus an organic coating was found to be best. The 50–50 solder was applied by metal spray gun after the surface had been thoroughly cleaned with carbon tetrachloride to remove grease, etc. followed by metal grit blasting at approximately 100 lb/sq.in. air pressure. Standard $\frac{1}{8}$-in round wire 40–50 solder was then applied by the metal spray gun to a thickness of from 0·03 to 0·04 in, determined by preliminary tests on steel plates. In addition to the solder and paint coatings described, cathodic protection is also used to minimize galvanic action. This is accomplished by using circular zinc plates 1 ft. in dia and $\frac{1}{2}$ in thick, mounted on the inside of each manhole door. It is expected that the solder coating will last at least two or three years.

Some of the Effects of Cadmium, Zinc, and Tin Plating on Springs. J. R. Gustafson. (Proceedings of the American Society for Testing Materials, 1947, vol. 47, pp. 782–802). Fatigue and salt-spray tests on 0·66% carbon steel springs electroplated with cadmium, zinc, and tin are reported. Cadmium gave the best results. Tests on springs heated to 400°, 500°, 600°, 700°, and 800° F indicated that cadmium could be used at up to 600° F. Zinc coatings broke down at 400° F, and tin gave poor results at all the test temperatures. R. A. R.

The Structure of Hot-Dip Zinc Coatings. (Metalloberfläche, 1947, vol. 1, June, p. 141). Methods of obtaining the desired appearance of zinc coatings are described.—R. A. R.

How Tank Capacity Affects Galvanizing Costs. W. G. Imhoff. (Products Finishing, 1948, vol. 12, May, pp. 42–54). The dimensions of a galvanizing kettle should be based on two factors: (1) If the heat applied to the tank sides exceeds 10,000 B.Th.U./sq.ft. the bath temperature will exceed 900° F which is the critical temperature above which rapid destruction of the kettle begins; and (2) from 15 to 20 lb of zinc should be allowed for each 1 lb of material to be coated. The sizes of kettles and pickling and rinsing tanks are discussed. R. A. R.

Progress of Fluxing in Hot Galvanizing Practice. A. T. Baldwin. (Steel, 1948, vol. 122, May 31, pp. 86, 88, 90). Wetting agents reduce surface tension in making up flux washes, and create foams which are stable when the flux is in fusion on molten zinc. The iron content of flux washes should be kept below 0·3 oz/gal; staining of pieces after a flux wash can be reduced by keeping them out of contact with one another.

Protecting the Surface of Steel by Copper Plating. H. Bröking. (Metalloberfläche, 1947, vol. 1, May, pp. 101–104; June, pp. 137–139). The plating of steel with copper by hot-rolling and the properties, including the adhesion, are described.—R. A. R.

The Deposition of Pure Boron. Part I. A Static Method for the Preparation of Boron Coatings. H. I. Schlesinger, G. W. Schaeffer, and G. D. Barbaras. (U.S. Atomic Energy Commission, 1944, No. MDDC 1338).

Electrolytically Prepared Tungsten–Cobalt, Tungsten–Nickel, and Tungsten–Iron Alloys. H. Offermanns and M. von Stackelberg. (Metalloberfläche, 1947, vol. 1, June, pp. 142–144). An electrolytic procedure for the deposition of tungsten–cobalt, tungsten–nickel, and tungsten–iron alloys containing up to 50% of tungsten is described. These alloys are very stable when attached by HCl, HNO_3 or H_2SO_4 of any concentration, but cannot be used for protecting other metals because a pore-free coating cannot be deposited. The atomic structure of the iron–tungsten alloy is body-centred cubic. R. A. R.

Aluminium-Coated Steel Strip. (Neuhausen Wochenbericht, 1948, No. 28, Feb. 10, p. 5: [Abstract] Mécanique Documentation, 1948, vol. 3, May, p. 2). Under the trade names "Feron" and "Triwallith," two German firms are producing aluminium-coated steel strip. This material, which is corrosion-resistant, is used instead of tinplate or galvanized iron, for the covering of cables.

Scuff- and Wear-Resistant Chemical Coatings. F. C. Young and B. B. Davis. (S.A.E. Quarterly Transactions, 1947, vol. 1, pp. 626–629, 661: Chemical Abstracts, 1948, vol. 42, May 20, col. 3303). Methods of applying and effects of caustic soda, iron oxide, and manganese/iron-phosphate treatments on cast-iron and steel parts are discussed.

Phosphate Treatments for Zinc Surfaces. E. E. Halls. (Metallurgia, 1948, vol. 37, Apr., pp. 299–305). Experiments on phosphate coatings applied by six methods are reported. Zinc castings, electro-zinc-plated steel components and galvanized steel panels were coated with a number of lacquers, enamels, and primers after phosphatizing and, for comparison, after chemical treatment by a chromate process or after no chemical treatment at all. Full details of the experimental results are recorded in 15 tables, but, while data concerning the finishing lacquers and enamels are given, the phosphate treatments are described only in general terms.—A. E. C.

Pre-Paint Treatment Combats Rust. R. F. Ayers. (Canadian Paint and Varnish Magazine, 1947, vol. 21, Mar., pp. 18, 20: [Abstract] Centre National de la Recherche Scientifique, Bulletin Analytique, 1948, vol. 9, No. 1/2, p. 251). The advantages of phosphatizing before painting as a protection against corrosion, are outlined.

Organic Coatings for Metals. E. E. McSweeney. (Plating, 1948, vol. 35, Apr., pp. 340–343). The characteristics, methods of application, and uses of synthetic-resin coatings, are outlined, and the use of plasticizers, solvents, pigments, and primers is discussed.—M. A. V.

Hard Facing by Metal Spraying. (Welding Engineer, 1948, vol. 23, Mar., pp. 62, 64: [Abstract] PERA Bulletin, 1948, vol. 1, June, p. 498). The article describes the newly developed "Sprayweld" process for applying thin layers of hard-facing alloy by means of a metallizing spray gun. The coating obtained is uniform and is firmly bonded to the base metal. The metallizing gun uses a flexible wire consisting of fine metallic particles embedded in a plastic binder,

the plastic being completely volatilized when the metal is sprayed. Fusion of the metal takes place at 1850°–1875° F.

Flame Spraying of Dust Extraction and Ventilation Plant. H. W. Greenwood. (Sheet Metal Industries, 1948, vol. 25, May, pp. 975–976). The advantages of flame spraying to produce a coating of either metal or plastics on fan blades, casings, ducts, and inside dust precipitation and separation plant are enumerated and some examples are described. R. A. R.

Metal Spraying. F. Loos. (Lastechniek, 1948, vol. 4, Apr., pp. 41–46). [In Dutch]. The metal-spraying pistol is described, and examples of the application of the process for corrosion- and wear-resisting purposes are discussed. R. A. R.

Improved Physical Properties of Titanium Enamels Provide Better Steel Finish. B. W. King. (Steel, 1948, vol. 122, June 21, pp. 108, 111, 128, 130). The preparation of titanium enamels is described and data on their resistance to impact, thermal shock, and to boiling in citric acid and caustic soda are presented.—R. A. R.

High Temperature Ceramic Coatings for Steel. (Steel Processing, 1948, vol. 34, May, pp. 245–247). The addition of alumina to porcelain enamel frits makes them very suitable for heat-resistant coatings for mild steels; a refractory of a higher thermal expansion is desirable for use on heat-resisting alloys.—J. P. S.

Mechanized Porcelain Enameling of Steel Tubs. F. M. Burt. (Products Finishing, 1948, vol. 12, May, pp. 14–26). An illustrated description is given of the plant and equipment for enamelling steel baths at the Norris Stamping and Manufacturing Co., Los Angeles, at which a high degree of mechanization has been achieved.—R. A. R.

Burning of Cast Iron during Enamelling and Its Relation to Blisters. T. W. Fenton. (Journal of the Canadian Ceramic Society, 1947, vol. 16, p. 39: British Ceramic Abstracts, 1948, Jan.–Feb., p. 29A). Blisters which cause rejection of vitreous enamelled ware are usually small, caused by the escape of gases which force their way through the enamel during firing, and so release their pressure. A number of precautions which can minimize blistering are enumerated, and the value is pointed out of the ground-coat, which is usually fired higher than the cover-coat so as to allow the gases to escape before the latter is applied.

Efficiency of Conveyor Drying and Enamelling Plants. A. M. Lehmann. (Ministry of Fuel and Power, Fuel Efficiency Committee Conference on "Fuel and the Future," London, Oct., 1946, pp. 263–266). Operating data for air-circulated direct-fired conveyor convection ovens for drying paints are presented and discussed, and the efficiencies of radiant- and convection-heated ovens are compared.—R. A. R.

Plastic Coatings Protect Components and Tools. (Machine Shop Magazine, 1948, vol. 9, May, pp. 94–96). Plasticized ethyl cellulose may be applied to tools by hot-dipping as a protective coating. Its properties are listed, and some uses illustrated.—M. A. V.

Action of Organic Sealing Coatings on Different Metals and Different Finishes. J. M. Perrone and J. P. Garvey. (Chimie des Peintures, 1947, vol. 10, Aug., p. 238: [Abstract] Centre National de la Recherche Scientifique, Bulletin Analytique, 1948, vol. 9, No. 3/4, p. 596). A résumé is given of a paper on the action of oleoresinous, alkyd, phenolic and vinylic coatings used for the protection of metals in the food industry (tinplate, aluminium, phosphatized iron, and black plate). The effect of pasteurization and sterilization is noted.

Protection of Polished Metal Sheets. (Plastics, 1948, vol. 12, No. 129, pp. 64–69, 84: [Abstract] PERA Bulletin, 1948, vol. 1, May, p. 433). This is a brief note on the plastic coating "Birlon." It can be applied by spray or roller or by dipping, and is easily removed by stripping. As well as providing protection against tarnish and scratching it may be used as a protective coating during forming operations.

Integrated Cleaning and Rust Proofing. (Steel, 1948, vol. 122, Apr. 12, pp. 90–92). The various processes in the manufacture of steel chairs for outdoor use are described, special reference being made to the rust-proofing with "Banox" (an amorphous phosphate coating).—R. A. R.

Tube Insulation and Corrosion. F. Hagman. (Teknisk Tidsskrift, 1948, vol. 78, May 22, pp. 341–345). [In Swedish]. Loss-of-weight corrosion tests were carried out on short lengths of steel tube 76 mm in outside dia which were cleaned and insulated with glass wool, slag wool, rock wool, cotton wadding, and two kinds of magnesia material; they were then immersed in water at 50° C for periods up to 260 days. Further tests were made on specimens insulated and wrapped with felt and other materials. The corrosion rate of unwrapped insulated tubes decreased with time, whereas for wrapped insulated tubes it increased. Tubes insulated with slag wool and cotton wadding corroded more rapidly than those insulated with rock wool and glass wool, and the corrosion was greatest on the magnesia-insulated tubes.—R. A. R.

Metallic Coatings and Surface Protection of Castings. M. Ballay. (Fonderie, 1946, Aug.–Sept., pp. 285–299). Metallic coatings may be divided into three groups according to their effect on the surface of the metal to be treated: (1) Coatings, such as painting, enamelling, and electolytic coatings, which do not modify the surface structure; (2) modification of the surface by cementation by absorption of nitrogen, aluminium, or zinc; and (3) modification of the surface by chemical or electrochemical reaction as in phosphatizing and anodic oxidation. The characteristics of each method of coating as they interest the foundryman, are outlined. J. C. R.

Apparatus for the Testing of Lacquer-Paint Coatings at Low Temperature. S. V. Yakubovich and T. I. Vorogushin. (Zavodskaya Laboratoriya, 1948, vol. 14, Apr., pp. 506–508). [In Russian]. An account is given of a method for testing properties of lacquer-paint coatings at temperatures down to − 60° C.

Service Trials of Painting Schemes Applied to a Steelworks Gantry. J. C. Hudson. (Journal of The Iron and Steel Institute, 1948, vol. 159, May, pp. 60–66).

Studies on Anti-Fouling Compositions. Part IV—The Relationship between Leaching Rate, Copper Loss, and Anti-Fouling Performance, under Raft and Service Conditions. H. Barnes. (Journal of The Iron and Steel Institute, 1948, vol. 159, June, pp. 175–185).

Electrolytic Resistance—a Means of Evaluating the Protection Qualities of Underwater Organic Coatings on Metals. R. C. Bacon, J. J. Smith, and F. M. Rugg. (Chimie des Peintures, 1947, vol. 10, Aug., pp. 237–238: [Abstract] Centre National de la Recherche Scientifique, Bulletin Analytique, 1948, vol. 9, No. 3/4, pp. 595–596). The measurement of electrolytic resistance has proved to be an excellent means of studying underwater coating. Log resistance/time curves enabled different factors affecting corrosion to be studied.

POWDER METALLURGY

The Pressing of Metal Powders. R. P. Seelig. (Australasian Engineer, 1947, Apr. 8, pp. 47–49). Theoretical and practical aspects of the pressing of metal powders are discussed. In the later stages of compression the compact approaches more and more the state of polycrystalline mass held together by pressure, friction, and adhesion. Factors affecting the design of the dies and the stroke of the press are also dealt with.—R. A. R.

Powder Metallurgy, Its Current Problems, Its Prospects. R. L. Ziegfeld. (Iron Age, vol. 161, Apr. 8, pp. 72–74). Some of the present applications of powder metallurgy are described, and the author concludes that a greater future will lie ahead of the industry when it has matured a little more.—J. P. S.

Powder Metallurgy as Applied to Permanent Magnets. A. E. Franks. (Iron Age, 1948, vol. 161, Apr. 8, pp. 82–85). The manufacture of permanent magnets by powder metallurgy from Alnico and similar alloys is described. Although these, in general, are small, it is sometimes preferable to manufacture large ones by this process, instead of casting them.—J. P. S.

Powder Metallurgy. R. Talmage. (Iron Age, 1948, vol. 161, Apr. 8, pp. 86–91). The author discusses the process from the point of view of rapid production and cost, concluding that much of the plant required is simple and inexpensive. J. P. S.

Sponge Iron and Iron Powder. J. F. Drapeau. (Iron Age, 1948, vol. 161, Apr. 8, pp. 92–94). The production of iron powder, suitable for powder metallurgy, from iron scale by way of a form of sponge iron is briefly described.—J. P. S.

The Electron-Microscope and the Metallurgy of Powders. P. Grivet, F. Bertein, H. Bruck, and J. Robillard. (Revue

de Métallurgie, Mémoires, 1948, vol. 45, Jan.–Feb., pp. 38–48). The authors first give a short account of the C.S.F. electron microscope and a note on electron diffraction. They then describe the microscopical analysis of powders, their preparation for examination, the use of metallic shadows, the determination of the statistical distribution of the different grain sizes, etc. Lastly, they deal with the examination of solid metallic surfaces, the plastic replicas used, and the increase of contrast by projecting metallic vapour at almost grazing incidence on to the replicas; typical electron micrographs of a number of metals are reproduced.—A. E. C.

Preparation of Metallic Powders by Fusion Electrolysis. J. L. Andrieux. (Société Française de Métallurgie, Mar. 27, 1947: Revue de Métallurgie, Mémoires, 1948, vol. 45, Jan.–Feb., pp. 49–59). If a molten salt is electrolysed at a temperature below the melting point of the metal produced, the latter will appear on the cathode in the form of small crystals. After a general discussion of the advantages of working at low temperatures, the separation of the crystals, the cathodic conditions and crystal size, the choice of electrolytic bath, and the mechanism of the process, the preparation of metal powders, including iron, nickel, cobalt, manganese, chromium, molybdenum, tungsten, titanium, binary metal-metalloid compounds, silicides, metallic phosphides, carbides, arsenides, antimonides and sulphides, and metallic alloys, is described in varying detail.—A. E. C.

PROPERTIES AND TESTS

What is Strength? J. B. Caine. (Foundry, 1948, vol. 76, July, pp. 80–81, 148–154; Aug., pp. 100–101, 166–170). The evaluation of the behaviour of metals, especially castings, by the tensile properties and behaviour in fatigue is considered. R. A. R.

The Testing of Engineering Metals by Deductive Methods. A. M. Armour. (Metropolitan Vickers Gazette, 1948, vol. 22, Mar., pp. 254–259). Non-destructive methods of testing metals and alloys are briefly reviewed in two sections, (*a*) testing for cracks and flaws, and (*b*) testing for correct material. R. A. R.

Electronics as Applied to Inspection and Testing. L. G. Ward. (Engineering Inspection, 1948, vol. 12, Summer Issue, pp. 24–31). Some applications of electronic methods for the determination of paint thickness and the detection of cracks are described.—J. P. S.

Elastic Properties in Tension and Shear of High-Strength Nonferrous Metals and Stainless Steel—Effect of Previous Deformation and Heat-Treatment. R. W. Mebs and D. J. McAdam, jun. (National Advisory Committee for Aeronautics, 1947, Mar., Technical Note No. 1100, pp. 1–105: [Abstract] Applied Mechanics Review, 1948, vol. 1, Apr., p. 111). Supplementing their previously published data with data from independent investigations, the authors present a complete résumé of their work on the influence of plastic deformation and of annealing temperature on the tensile and shear elastic properties of nickel, Monel, Inconel, copper, 13/2 and 18/8 chromium–nickel steels, and aluminium–Monel. The relationship between hysteresis and creep is first obtained by repeated cyclic stressing of annealed stainless-steel specimens over a constant load range, as an aid in devising a loading-time schedule to be used in measurements of elastic properties. The authors point out that the time schedule is very important in such measurements, and that in the present tests the measurements were made only after holding each load for a period of two minutes, thereby allowing the positive and negative creep to reach low rates so that accurate and sensitive-set measurements could be made. Use is made of stress-strain curves to derive the variation of secant modulus with stress, and of stress-set curves to derive five proof-stresses used as indices of the elastic or yield stress. The authors point out that both the stress-strain and stress-set relations are necessary for a complete representation of the tensile elastic properties of a metal. The variation of the proof stress and modulus of elasticity with plastic deformation or annealing temperature is explained in terms of relative dominance of internal stress, lattice-expansion or work hardening, and crystal reorientation. Considerable attention is also directed to the variation of Poisson's ratio with plastic deformation and annealing temperature.

The Measurement of Large Forces. P. Nycander. (Instruments and Measurements Conference, Stockholm, 1947: Statens Provningsanstalt, 1948, Report No. 102). The method used at the State Material Testing Station in Stockholm for the measurement of forces up to 500 tons in compressive testing is described.—R. A. R.

Plastic Deformation and the Lattice Parameter. B. M. Rovinski and T. V. Tagoonova. (Journal of Technical Physics, U.S.S.R., 1947, vol. 27, Oct., pp. 1137–1142: [Abstract] Applied Mechanics Review, 1948, vol. 1, Apr., p. 109). The authors investigated the influence of plastic deformation on the change of lattice parameter of Armco iron, copper, and two copper alloys, one containing 19·92 atomic per cent of zinc, the other 17·4 atomic per cent of aluminium. To eliminate the influence of the residual stress on the lattice parameter the experiments were performed on powders obtained by filing, rather than on plastically deformed blocks of metal. The influence of cold working was then gradually removed by heating the filings to various temperatures until full recrystallization was achieved. X-ray-diffraction back-reflection diagrams were taken in all cases, but the lines were too diffuse to permit lattice parameter determination for the "as filed" condition and for the heat-treatment at the lowest temperature. However, considerable line-broadening still was present for other treatments below the recrystallization temperature. It was therefore concluded that for these treatments the grain still was plastically deformed. Despite this circumstance the measurements failed to reveal change in lattice parameter greater than the probable curve (0·01%) due to plastic deformation. On the basis of these experiments it was concluded that the change of lattice parameter reported by Wood and Burgers was produced by the residual stress and not by plastic deformation. On the other hand, changes reported by Schafer for 50 atomic per cent iron–aluminium alloy may be due to phase changes (from ordered to disordered state) produced by plastic deformation.

On the Restricted Applicability of the Principle of Least Work in the Plastic Domain. P. P. Bijlaard. (Proceedings of the Koninklijke Nederlandsche Akademie van Wetenschappen, 1947, vol. 50, No. 4, pp. 397–405: [Abstract] Applied Mechanics Review, 1948, vol. 1, Apr., p. 109). This paper deals with variations of the elastic and plastic strain energies of a yielded strip of a steel plate. The background of the work presented is contained in several previous papers by the same author and many references are made to this former work. Emphasis is placed upon the transitional domain between the elastic and the plastic region. During this transition the state of stress must change from one corresponding to minimum elastic strain energy to one corresponding to minimum shear strain energy. Proof is given that the total strain energy, *i.e.*, elastic plus plastic, does not in general acquire extreme values in this transitional domain and hence each component must be considered independently. Different conditions under which the strain energy can be varied and the restricted use of the results of the variations in terms of possible applications are discussed.

Effect of Wetting. Influence of Non-Corrosive Liquids on the Resistance to Rupture of Solid Bodies. C. Benedicks. (Journées de la Corrosion, Oct. 8, 1947: Revue de Métallurgie, Mémoires, 1948, vol. 45, Jan.–Feb., pp. 9–18). Bending fatigue tests were made in the Chevenard micromachine on several materials to examine the influence of wetting the surface on the fatigue strength. In the case of steel (carbon about 0·7%, silicon 0·3%, manganese 0·4%) water, ethyl alcohol, and octyl alcohol reduced the fatigue strength, while 10% NaOH solution, petroleum, benzene, and other liquids, including some corrosion protectives raised it. Toluene and other liquids were without effect. The author discusses the theory of these phenomena and considers them in relation to others, such as corrosion fatigue, caustic embrittlement, season cracking, weld-brittleness, etc.—A. E. C.

The Static Hardness-Testing Processes. E. Vrana. (Österreichischer Maschinenmarkt und Eisenhandel, 1948, vol. 3, May, pp. 120–125). The Brinell, Vickers, and Rockwell methods of hardness testing are reviewed and the relationship between test data by the three methods is shown. A brief description of the universal method proposed by W. Hengemühle, which is based on the Vickers principle, is given. R. A. R.

Micro-Hardness Testing. (Scientific American, 1947,

vol. 176, June, pp. 265–266: [Abstract] Centre National de la Recherche Scientifique, Bulletin Analytique, 1948, vol. 9, No. 5–6, p. 731). By means of a diamond pyramid set at an angle of 136° it was possible with a very light load to make impressions that could be examined under the microscope. This method can be used on crystals isolated in the mass.

Micro-Hardness Testing of Metals. E. W. Taylor. (Journal of the Institute of Metals, 1948, June, vol. 74, pp. 493–500). Reference is made to the methods normally employed for testing the hardness of metals and alloys, and attention is directed to the fact that when an indenter is used the impressions made are usually large in relation to the microstructure of the specimen. The desirability is stressed of a form of hardness test which can be applied to a particular crystal or to a small selected area, and the factors controlling the application of very light loads to a diamond indenter are examined.

A new microhardness tester is described, the design of which closely follows the principles discussed earlier. A number of examples are given of microhardness tests variously applied to small selected areas of particular specimens.

Design and Application of Supersonic Flaw Detectors. D. C. Erdman. (Transactions of the American Institute of Electrical Engineers, 1947, vol. 66, pp. 1271–1276). General parameters are given for the circuits that make up echo-type supersonic flaw detectors used in connection with large forgings. The applications of such equipment are outlined and a bibliography of the literature on the subject is included. R. E.

Measurement of Damping by the Vibration Method. V. A. Zhuravlev. (Zavodskaya Laboratoriya, 1948, vol. 14, May, pp. 614–620). [In Russian]. An account is given of experiments in which internal friction effects were investigated by a study of the damping of a vibrating specimen. The cylindrical specimen was suspended at two nodal points and vibrated by an electromagnetic exciter, the amplitude of the vibrations being measured microscopically and electrically for large and small amplitudes respectively. This arrangement was used to investigate the relation between the amplitude and the decrement, and a mathematical discussion of this relationship is given. The influence on the decrement of the following factors was also studied: (1) Frequency of vibration; (2) mechanical working; (3) heat-treatment (for a 0·45% carbon steel and an alloy steel with 0·37% of carbon, 1·45% of nickel, 1·3% of chromium, and 0·3% of molybdenum); and (4) fatigue. Results are shown graphically.—S. K.

Fundamental Investigation of Graphitization of Piping. L. E. Hankison, T. E. Purcell and A. D. Bailey. (Edison Electric Institute Bulletin, 1948, vol. 16, Feb., pp. 43–44: Heating, Piping and Air Conditioning, 1948, vol. 20, Apr., pp. 76–77: [Abstract] Metals Review, 1948, vol. 21, May, p. 12). Results obtained at Battelle Memorial Institute since March, 1947, involving continued tests at 1125° F of previously tested materials which had indicated good resistance to graphitization, and further efforts to evaluate the influence of elastic stress and plastic deformation on graphite formation, are given. The alloys investigated were as follows: Plain carbon low-aluminium steel; carbon–molybdenum low-aluminium steel G 18; carbon–molybdenum low-aluminium steel G 19; carbon–molybdenum silicon-deoxidized steel; carbon–molybdenum titanium-deoxidized 0·5–2·5% chromium steel; 0·5–1% molybdenum steel; and molybdenum–vanadium steel.

Graphitisation of Welds in Steam-Pipes Made of Molybdenum Mild Steel. F. Meunier. (Revue de la Soudure, 1947, vol. 2, pp. 51–71: British Abstracts, 1948, BI, May, col. 206). Breakdown of welded joints in high-pressure pipelines after several years of use was found to be due to a change in the structure of the metal immediately surrounding the seam caused by separation of the carbon of the steel in the form of irregular chains of graphite. The causes of this graphitization have been investigated by studying the change of mechanical properties and structure with time of heating at temperatures between 300° and 1200°. The presence in the steel of manganese, chromium, vanadium, and molybdenum delays breakdown of the joint, whilst silicon, aluminium, titanium, copper, and nickel accelerate graphitization. Curative and preventive measures are suggested, including heat-treatment to redissolve the graphite. Photomicrographs of the various steels and the change which occurs on graphitization are included.

Dilatometry. N. A. McKinnon. (Australian Institute of Metals, Symposium on Recent Advances in Physical Metallurgy: Australasian Engineer, 1948, Mar. 8, pp. 60–69). *See* Journ. I. and S.I., 1948, vol. 159, July, p. 337.

The Influence of Niobium and Titanium on Stainless Steel. A. Yaskevitch and A. Samarine. (Bulletin de l'Académie des Sciences, U.R.S.S., Classe des Sciences Techniques, 1946, No. 4, pp. 593–602: [Abstract] Centre de Documentation Sidérurgique, Bulletin Analytique, 1948, vol. 5, Feb., p. 47). The authors have studied the influence of niobium on the strength of metal, and note that it increases resistance to intercrystalline corrosion and does not diminish resistance to chemical corrosion. It does not affect tensile strength and only slightly reduces elongation value.

Nickel Alloys. A. E. Williams. (Mining Journal, 1948, Annual Review Number, Apr., pp. 39–47). The mechanical properties of a number of nickel-bearing materials are given, including nickel-clad steel, nickel cast iron, and case-hardened nickel steel.—M. A. V.

The Influence of Small Contents of Lead on the Ability to Forge and Roll Austenitic Stainless Steels. S. Bergh. (Jernkontorets Annaler, 1948, vol. 132, No. 6, pp. 213–220). [In Swedish]. The effect of small amounts of lead on the hot-working properties of 18/8 and 25/20 stainless steels was investigated. To obtain good "hot-workability" the lead content must not exceed about 0·005%. Liquid steel is able to dissolve some tenths of 1% of lead, but in the solid state the solubility is much less. Spectrographic analyses and lead prints have revealed lead inclusions in the primary grain boundaries which may be the cause of the reduced hot-workability. The high rolling pressure may cause fracture along the grain boundaries where cohesion is reduced by the presence of lead.—R. A. R.

Boron-Treated Steels. (National Bureau of Standards: Industrial Heating, 1948, vol. 15, Apr., pp. 624–630). *See* Journ. I. and S.I., 1948, vol. 159, July, p. 342.

The Effect of Silicon on the Properties of Cast Carbon and Carbon Molybdenum Steels. N. A. Ziegler, W. L. Meinhart, and J. R. Goldsmith. (Transactions of the American Society for Metals, 1948, vol. 40, pp. 617–648). *See* Journ. I. and S.I., 1948, vol. 158, Apr., p. 540.

Influence of Copper in Cast Iron. (Fonderie, 1946, June, pp. 230–231). Brief notes are given on the influence of copper on the structure and mechanical properties of cast iron, and its hardening power compared with that of other elements. J. C. R.

Alloying Elements' Effect on Tensile Properties of Malleable Iron. H. A. Schwartz, and W. K. Bock. (American Foundryman, 1948, vol. 13, Apr., pp. 130–133). Insofar as phosphorus, silicon, manganese, and chromium dissolve in ferrite they strengthen the matrix of malleable cast iron; their cumulative effect can be calculated by assuming silicon and chromium to be of equal potency, phosphorus as equivalent to 6·5 times the same weight of silicon, and manganese as equivalent to 0·5 times the same weight of silicon, but the effect decreases with the amount of the alloy present.—J. P. S.

Selection of Engineering Materials for Coke Plants. C. F. Pogacar. (Eastern States Blast Furnace and Coke Oven Association: Blast Furnace and Steel Plant, 1948, vol. 36, Apr., pp. 439–442; May, pp. 555–559: Steel, 1948, vol. 122, June 7, pp. 111, 114–126). This is a general description of the properties of metals and their resistance to fatigue, creep, wear, and corrosion, with some reference to the conditions occurring in coke-oven plants.—R. A. R.

Inspection of Pressure Vessels and Tanks. A. P. Maradudin. (Western Metals, 1948, vol. 6, Apr., pp. 23–26).

Effect of Variables in Welding Technique on the Strength of Direct-Current Metal-Arc-Welded Joints in Aircraft Steels. Part III. Repeated-Stress of Joints in S.A.E. 4130 Seamless Steel Tubing. C. B. Voldrich and E. T. Armstrong. (National Advisory Committee for Aeronautics, 1948, Technical Note 1262: [Abstract] Metals Review, 1948, vol. 21, June, p. 56). Tests were made on joints in S.A.E. 4130 seamless steel tubing. Variables included type of electrode, speed of welding, current, position, amount of preheat, and other factors which in turn controlled the weld contour, penetration, and depth of the heat-affected zone. A range of joint design was also investigated. Fatigue data showed that stress concentrations

due to weld geometry were the predominating factors in determining endurance life. Previous concepts relative to the behaviour of fillet and butt welds were confirmed.

Modern Cast Irons : Application—Properties—Manufacture. A. M. Ondreyco. (Western Machinery and Steel World, 1948, vol. 39, Apr., pp. 90–93).

Physical Properties of Steel Control Cable with Nearly the Same Expansion Rate as Aluminium Alloy. C. W. Meyers. (Automotive and Aviation Industries, 1946, vol. 95, Dec. 15, pp. 44–45, 62, 64 : [Abstract] Centre de Documentation Sidérurgique, Bulletin Analytique, 1948, vol. 5, Feb., p. 49). The composition and mechanical and magnetic properties of a steel developed during the war for the manufacture of flexible cables, are given. It meets all requirements within the temperature range $+40°$ C to $-55°$ C.

Influence of Rolling and Forging on the Mechanical Properties of Carbon Steel. M. V. Rastegaev. (Bulletin de l'Académie des Sciences, U.R.S.S., 1948, No. 1, pp. 103–107).

Some Developments in High-Speed Steels. J. Jacquet. (Pratique des Industries Mécaniques, 1947, vol. 30, July, pp. 218–220 : [Abstract] Centre National de la Recherche Scientifique, Bulletin Analytique, 1948, vol. 9, No. 1/2, p. 252). The following aspects are reviewed : The history of high-speed steels ; American molybdenum steels ; advantages appertaining to molybdenum steels and precautions that have to be taken to avoid oxidation ; and heat-treatment processes.

High Temperature Bolting Materials. E. L. Robinson. (American Society for Testing Materials, 1948, Preprint No. 16s). This report presents data on the creep strength and other physical properties of a large number of alloy steels for bolts for service at temperatures up to 1500° F.—R. A. R.

Hardened Cast Iron Alloys for Ball-Mill Linings. V. G. Makh. (Ogneupory, 1947, vol. 12, p. 86 : British Ceramic Abstracts, 1948, Mar.–Apr., p. 77A). Hardened cast-iron alloy linings are less durable than manganese steels, but are equal to carbon steels in performance. They can be manufactured more cheaply than manganese steel linings. The composition and type of structure giving the best results, and the casting technique giving such a structure, are described.

Ageing of Soft Steels Alloyed with Aluminium. L. Szalay. (Bányászati és Kohászati Lapok, 1946, vol. 1, pp. 40–42 : Chemical Abstracts, 1948, vol. 42, Apr. 20, col. 2562). For the manufacture of articles that will be subjected to increasing tensile stress soft steels alloyed with aluminium and containing 0·15–2·5% of carbon are better than lower carbon steels since ageing appears later.

Surface Analysis with Plastic Replicas. A. T. Chisholm and J. M. Lickley. (Engineering, 1948, vol. 165, Apr. 23, p. 389). A technique is described for the preparation and mounting of plastic replicas of machined surfaces for the measurement of surface roughness. Details are given of a method of mounting cellulose acetate replicas which prevents the formation of extraneous waviness ; the method has been very successful for measurements of thrust-bearing faces and journals of large turbine spindles, and tooth flanks of gear wheels.—R. A. R.

Modern Engineering Cast Iron. D. J. Reese. (Metal Progress, 1948, vol. 53, Apr., pp. 539–544). Developments in the properties of cast iron are reviewed. It is looked upon as an iron–carbon–silicon alloy and the influence of the form in which the carbon is presented is pointed out. Examples of the outstanding performance of cast-iron housings on tractor transmissions, cast-iron rolls in a tube mill, and cast-iron crankshafts are cited.—R. A. R.

Considerations in the Fabrication of Stainless Steel. A. S. Tuttle. (Canadian Metals and Metallurgical Industries, 1948, vol. 11, Feb., pp. 18–22, 36, 35). The characteristics of the martensitic, ferritic, and austenitic stainless steels are explained, and the properties and fabrication of the austenitic steels are discussed ; a table of the properties of steel 302, 303, 304, 309, 310, 316, 321 and 347 of the American Iron and Steel Institute austenitic steels is presented.—R. A. R.

METALLOGRAPHY

Heterogeneity of Steel Ingots and Their Crystallization. V. A. Davidenkov. (Bulletin de l'Académie des Sciences, U.R.S.S., Classe des Sciences Techniques, 1947, Dec., pp. 1683–1703 [in Russian] : [Abstract] Metals Review, 1948, vol. 21, May, p. 12). Factors which indicate directly or indirectly the presence of heterogeneity were investigated on several ingots produced by different methods. Relationships between content, number, and dimensions of inclusions ; mechanical properties ; toughness ; and microstructures were established. Conclusions concerning the mechanism of crystallization are given.

Atomic Structure of Engineering Metals. S. K. Ghaswala. (Science and Culture, 1947, vol. 12, pp. 388–395 : Chemical Abstracts, 1948, vol. 42, June 10, col. 3711). This is a review, with forty-seven references, on the relation of atomic structure to elasticity, plasticity, yield, creep, fatigue, strain-hardening, and compressibility.

Method of Investigation of Rapid Processes. M. E. Blanter. (Zavodskaya Laboratoriya 1947, vol. 13, pp. 233–235 : Chemical Abstracts, 1948, vol. 42, May 20, col. 3298). Fast isothermal transformations of steels at supercritical temperatures are investigated on very thin samples (foils 0·1–0·2 mm thick, 5 × 15 mm) with the aid of an electron tube-controlled electromagnetic device permitting immersion of the sample into a constant-temperature furnace for time intervals from 0·2 sec up, controlled within 0·01 sec. Cooling is done in a stream of CO_2. The lag due to the time necessary for the sample to reach the temperature of the furnace constitutes 0·035–0·06 sec. The method is illustrated by the curve of hardness of a steel (carbon 1·05%, chromium 1·42%) in the isothermal transformation of pearlite into austenite at 820°, plotted against length of heating ; constant maximum hardness is reached after 10 sec.

The Classification of the Solubilities of Elements in Iron. II—Continuous Solid Solutions of Iron. I. I. Kornilov. (Bulletin de l'Académie des Sciences de l'U.R.S.S., Classe des Sciences Chemiques, 1947, pp. 337–346. [In Russian] : Chemical Abstracts, 1948, vol. 42, May 20, col. 3299). Chemical elements having the same crystal lattice as iron and differing in atomic diameter by not more than 8% can form a continuous series of binary solid solution with iron. Since iron can form solid solutions based on the α-iron lattice (ferrite) or on the γ-iron lattice (austenite), the eight elements that can form continuous binary solid solutions with iron are divided in this manner. Chromium and vanadium are the only elements that form continuous binary solid solutions of the ferrite type. Since it has been reported that there is complete solid solubility in the chromium–vanadium system, it is supposed that the ternary system iron–chromium–vanadium shows continuous solid solubility. A hypothetical diagram is shown with the solid-solution region broken by the γ loop and by the compounds FeV and FeCr. The elements cobalt, nickel, rhodium, palladium, iridium, and platinum form continuous solid solutions of the austenite type. A total of 15 ternary continuous solid-solution systems of this type is possible. The solid solution in each of these ternaries is interrupted by the $\alpha \rightleftharpoons \gamma$ transformation, and in all, except possibly the iron–nickel–cobalt system, chemical compounds form in the solid state. The mode of occurrence of both of these discontinuities in the solid solution is shown schematically for the iron–nickel–platinum system. A total of 20 quaternary austenitic continuous solid-solution systems involving iron is possible, and a tetrahedral representation of the $\alpha \rightleftharpoons \gamma$ transformation in a typical system is given. Eleven quinary systems are possible. There are twenty references.

Method of Phase (Carbide) Analysis of Steel. S. M. Gutman. (Zavodskaya Laboratoriya, 1947, vol. 13, Dec., pp. 1403–1412). [In Russian].

Method for Determination of the Temperature Range of the Ac_1 Transformation in Steels. I. N. Golikov. (Zavodskaya Laboratoriya, 1947, vol. 13, Dec., pp. 1435–1440). [In Russian].

The Influence of Ingot Heterogeneity on the Kinetics of Austenite Decomposition. N. P. Chuprakova and V. D. Sadovskii. (Stal, 1948, No. 3, pp. 262–263). [In Russian]. A brief account is given of experiments in which magnetometric methods were used to follow the decomposition of austenite in specimens taken from different parts of an ingot ; the results indicate that the rate of this decomposition varies considerably with location in the ingot.—S. K.

Determination of Phase Fields. W. L. Fink. (Metal Progress, 1948, vol. 53, Apr., pp. 530–535). Several ways of obtaining data for the construction of constitutional diagrams and their advantages and disadvantages are described and

discussed. The methods are: Thermal analysis; with the microscope; X-ray techniques including powder, back-reflection, and radiography; measurements of electrical resistivity and changes in volume; electrochemical methods; and the chemical analysis of phases.—R. A. R.

Notes on Equilibrium Diagrams. L. Grenet. (Echo des Mines, 1948, May, pp. 95–96; June, pp. 119–120). Observations that can be deduced from the study of equilibrium diagrams are discussed with special reference to the iron–carbon diagram.—R. F. F.

CORROSION

Fundamentals of Corrosion Control. (Electrical World, 1947, vol. 128, No. 7, pp. 86–87: [Abstract] Corrosion, 1948, vol. 4, July, p. 28). Factors which stimulate corrosion are listed, including the following: Damp atmosphere, oxygen dissolved in water film, acids, salts, contact of dissimilar metals, presence of depolarizing surface, non-uniformity of the metal. Corrosion due to electrolysis and the use of alloying metals to make the basis metals slightly cathodic are discussed very briefly. Diagrams illustrating the principle of the electromotive series are included.

A New Atmospheric Corrosion Laboratory Test for Ordinary Steels with Low Alloy Additions. E. Herzog. (Métaux, Corrosion, Usure, 1945, vol. 20, Apr., pp. 47–51). An apparatus is described in which specimens were exposed in a humid atmosphere so that small drops of water were condensed on them and rusting rapidly occured. Tests were then carried out to note: (1) The scatter of results with changes in the location of specimens; (2) the influence of humidity; (3) the influence of small additions of copper, chromium, aluminium, phosphorus, sulphur, and of surface condition; (4) the difference between the classification obtained by this test and that obtained by exposing specimens in an industrial atmosphere. The results are presented in tabular form. J. C. R.

Surface Condition and Passivation. L. Guitton. (Métaux, Corrosion, Usure, 1945, vol. 20, May, pp. 68–69). A brief account is presented of experiments carried out to determine the effect of various surface conditions on the degree of passivation acquired by stainless steels.—J. C. R.

Stabilization of Austenitic Stainless Steel. S. J. Rosenberg and J. H. Darr. (Journal of Research of the National Bureau of Standards, 1948, vol. 40, Apr., pp. 321–338). A study was made of the resistance to intergranular attack of twenty-three 18/10 chromium–nickel austenitic steels in twelve different initial conditions. Susceptibility to intergranular attack was determined after seven sensitizing treatments, followed by exposure for a maximum of 14 days in a boiling acidified copper sulphate solution. It was found that maximum susceptibility to intergranular attack was developed by sensitizing either 8 or 21 days at 1020° F. The straight carbon austenitic steels were quite vulnerable to attack although decrease in carbon decreased the vulnerability. The columbium- and titanium-treated steels were satisfactorily resistant to attack provided the Cb/C or Ti/C ratios were sufficiently high. These ratios varied, depending on the initial condition of the steel. The carbon content of the treated steels had no influence upon the resistance to intergranular attack, the predominating factor being the Cb/C or Ti/C ratio.

Three of the Eight Forms of Corrosion. M. G. Fontana. (Metal Progress, 1948, vol. 53, Feb., pp. 231–233). The author distinguishes between the following forms of corrosion: (1) Uniform attack or general corrosion; (2) intergranular corrosion; (3) pitting; (4) dezincification; (5) erosion-corrosion; (6) galvanic or two-metal corrosion; (7) stress-corrosion; and (8) concentration cell corrosion. The first three forms are described with micrographs illustrating the second and third forms.

Galvanic Corrosion and Concentration Cell Corrosion. M. G. Fontana. (Metal Progress, 1948, vol. 53, Mar., pp. 382–386). This paper deals with two more of the eight forms of corrosion (*see* preceding abstract). These are galvanic and concentration-cell corrosion. The theory of these and the principles of cathodic protection are explained.—R. A. R.

Stress Corrosion and Corrective Measures. M. G. Fontana. (Metal Progress, 1948, vol. 53, June, pp. 838–840). This paper deals with the last of the eight forms of corrosion (*see* preceding abstracts). Many examples of stress-corrosion cracking are cited and some methods of preventing it, particularly improvement of design, are discussed.—R. A. R.

Intergranular Corrosion of Cast Austenitic Rustless Steels. H. A. Pray. (Journées de la Corrosion, Oct. 8, 1947: Revue de Métallurgie, Mémoires, 1948, vol. 45, Jan.–Feb., pp. 19–31). The intergranular corrosion of thirty alloy steels with chromium varying from about 15% to 22%, nickel from about 7% to 11%, and carbon from about 0·05% to 0·30% (manganese about 0·8%, silicon about 1%, nitrogen about 0·06%) was investigated by boiling specimens in 65% nitric acid for 2 to 5 periods of 48 hr each, the weight and electrical resistance being noted before and after each period. The influence of heat-treatment on the intergranular corrosion was also examined. Susceptibility to corrosion was due to precipitation of carbides in the primary grain boundaries; if the precipitation occurred in the ferrite phase there was no susceptibility, so that the presence of ferrite within the austenite matrix could suppress the tendency to intergranular corrosion, because the carbides precipitated preferably in the ferrite. By adjusting the chromium, nickel, nitrogen, silicon, and carbon contents, enough ferrite could be made to form to reduce or even suppress the susceptibility to intergranular corrosion engendered by subsequent treatment at temperatures at which carbides would precipitate. In these alloys ferrite has a bad effect on the forging and rolling properties, hence the great care taken to ensure a completely austenitic structure in materials to be worked; in the cast condition, on the other hand, the amount of ferrite needed to suppress the susceptibility has no noxious effect—in fact it seems to be a desirable constituent. Susceptibility to intergranular corrosion can cause incomplete solution of the carbides or defective heat-treatment. The minimum temperature to dissolve carbides depends on the carbon content and varies from 760° to 1260° C and more.—A. E. C.

Contribution to the Study of the Intercrystalline Corrosion of Steel by Nitrates. M. Smialowsky. (Journées de la Corrosion, Oct. 8, 1947: Revue de Métallurgie, Mémoires, 1948, vol. 45, Jan.–Feb., pp. 32–37). In the first experiments described, the influence of very small additions of various acids, bases, and other chemicals on the corrosiveness of boiling ammonium and sodium nitrates was examined. The specimens were 0·06% carbon steel wires and were held under a tension of 10 kg/sq.mm.; the time taken for rupture to ensue was used as a measure of the corrosiveness. In the ammonium nitrate solution, small additions of nitric acid and caustic soda had little effect, phosphoric and arsenious oxide delayed corrosion sensibly, manganese and permanganate ions accelerated corrosion, even more so in sodium nitrate solution.

In the second group of experiments boiling 65% ammonium nitrate was flowed over specially shaped horizontal tensile test-pieces, cut from 1-mm. sheet, while under different loads. In three of the steels the carbon was about 0·03–0·06%, manganese about 0·28%, copper 0·16% and aluminium 0·016%; the fourth steel, deoxidized with aluminium, contained carbon 0·18%, manganese 0·57%, copper 0·09%, and aluminium 0·054%. The specimens were tested in either the hot-rolled or the annealed condition. The time required for fracture was plotted against the load applied expressed as a percentage of the tensile stress of the metal. The grain-size was also measured. In the first three steels there was no direct relation between the rate of corrosion and the grain-size. The method of applying the corroding liquid and the testing temperature greatly affected the results. The fourth steel in the hot-rolled condition offered considerable resistance to corrosion, but when annealed for ½ hr in air at 930° C and furnace-cooled its resistance decreased to the value of the other three steels. Coarsely polished specimens of the last steel broke more quickly than those more finely polished.—A. E. C.

Surface States and the True Physico-Chemical Properties of Metals. G. Chaudron. (Journées de la Corrosion, Oct. 8, 1947: Revue de Métallurgie, Mémoires, 1948, vol. 45, Mar.–Apr., pp. 61–67). The influence of the state of the surface on the corrosion and oxidation of several metals including iron is discussed. Two graphs show the effect on the oxidation of iron at 850° C of the degree of polishing applied in comparison with passivation in nitric acid and of the thickness of the zone disturbed by the polishing.—A. E. C.

Influence of the Texture and of the Crystalline Orientation on the Tendency to Dry Oxidation of Copper and of Iron. J.

Bénard and J. Talbot. (Journées de la Corrosion, Oct. 8, 1947 : Revue de Métallurgie, Mémoires, 1948, vol. 45, Mar.–Apr., pp. 78–85). The influence of previous mechanical working by rolling and by stretching and of the crystalline orientation on the rate of oxidation of copper and raised temperatures and the structure of the oxide film formed on the copper were investigated. The results obtained are described and discussed in relation to results yielded by earlier experiments on iron in order to explain certain features observed in regard to the latter metal. The influence of the surface state on the rate of oxidation is also dealt with.—A. E. C.

Cracking of Mild Steel by Stress-Corrosion. J. T. Waber and H. J. McDonald. (Journées de la Corrosion, Oct. 8, 1947 : Revue de Métallurgie, Mémoires, 1948, vol. 45, Mar.–Apr., pp. 93–104). According to the theory here propounded, stress corrosion is due to the continued and combined effects of precipitation and stress ; precipitation puts the metal in a condition to become fissured by corrosion, the fissures in the surface set up high local stresses, and then the latter accelerate precipitation. More precisely, the mechanical stresses speed up the separation of the new phases from the initial supersaturated solid solution, and one of these phases with the initial phase form a galvanic cell. The formation of the new product at the grain boundaries and its solution proceed continuously ; fissuration therefore follows an intercrystalline path. A deduction from this theory is that dissolved nitrogen, free to form nitride of iron, is responsible for fissuration ; removal of nitrogen (by various means described ; treatment with hydrogen, with aluminium or titanium) reduces fissuration, and with its reintroduction, fissuration reappears. The retarding effect of carbon is discussed also. A. E. C

Effect of Atmospheric Corrosion on Maintenance and Economics of Overhead Line Hardware and Guy Strand. C. J. Couy. (Corrosion, 1948, vol 4, Apr., pp. 133–140 ; May, pp. 207–218 : June, pp. 287–303). This paper deals generally with the problems of corrosion in connection with overhead lines. Part I deals with the relative corrosion of various types of steel, and the value of four types of metallic coatings, *e.g.*, zinc, copper, lead, and aluminium, in an industrial atmosphere, such as that of Pittsburgh, Part II deals with the effect of corrosion on the structural strength of overhead line parts. the evaluation of service life, and the annual inspection cost. Part III presents tables and curves of corrosion data of different sizes of galvanized guy wires ; the relationships corrosion/time and strength/time are shown. R. A. R.

An Investigation of the Corrosive Effects of Soils on Steel Plates. G. D. Gemmell. (New Zealand Engineering, 1948, vol. 3, May 10, pp. 468–469). In 1943 the Corrosion Committee of the New Zealand Institution of Engineers arranged for sets of 12 mild-steel specimens 6 × 3 × 10-in gauge to be buried at different places in New Zealand. Plates buried at Auckland, Wellington, and Christchurch have been exhumed and this report presents data on the nature of the soil and the loss in weight.—R. A. R.

On the Formation of Calcium Carbonate and Silica Deposits during the Corrosion of Iron in Soft Water. E. Herzog. (Métaux, Corrosion, Usure, 1945, vol. 20, May, pp. 57–60). Results are presented of tests undertaken to examine the conditions in which calcium carbonate and silica deposits form on mild steel immersed in soft water. Particular attention is paid to the influence of the surface condition of the steel and of the circulation rate of the water.—J. C. R.

Mechanism of Corrosion of Water Pipes. T. M. Riddick. (Water & Sewage Works, 1947, vol. 94, July, pp. R-149–R-154 : [Abstract] Corrosion, 1948, vol. 4, July, p. 42). The above mechanism is discussed and three empirical formulæ, derived to express numerically the corrosive tendencies of water from analyses, are presented.

The Problem of the Corrosion of Underground Piping. (Métaux, Corrosion, Usure, 1945, vol. 20, Mar., pp. 37–42). This article comprises the communications of three authors. In the first P. Bidart deals with the occurrence of electrolytic corrosion of buried pipes and means of preventing it. Reference is made to stray currents from tramway lines. In the second M. Maurin discusses chemical, electrochemical, and electrolytic corrosion and their causes, and means of protecting the piping attacked. The third paper by M. Junière is mainly concerned with the corrosion of aluminium piping.—J. C. R.

The Corrosive Conditions in the Exhaust Pipes of Internal Combustion Engines. G. Schikorr. (Metalloberfläche, 1948, vol. 2, Apr., pp. 74–79). After discussing the composition and corrosive properties of the exhaust gases from internal combustion engines an apparatus is described with which the corrosion of metals in a continuous stream of exhaust gas at 400°, 180°, and 70° C could be measured by weight-loss determinations. Tests were made on cast iron and copper. With regard to cast iron it was observed that : (*a*) There was moderate attack at 400° C, but after one day the rate of corrosion fell to about 0·06 mm/annum ; (*b*) at about 180° C the attack was so slight that it could not be determined ; and (*c*) at 70° C there was considerable corrosion, the rate probably reaching about 1·4 mm/annum. The significance of the results in relation to the material used for pipe systems for the exhaust-gas heating of rooms is discussed.—R. A. R.

Electrical Deaeration Licks Corrosion in This Plant. V. Rodwell. (Power, 1947, vol. 91, May, pp. 92–94 : [Abstract] Corrosion, 1948, vol. 4, July, pp. 92–94). Laboratory tests showed such promising results that the power-plant deaerators at Slough Estates, England, were redesigned and electrodes installed so that boiler feedwater could be treated electrically. Basic treatment, corrective measures, test results, plant application, process operation, and operating results are discussed.

Corrosion Research in Union-Castle Ships. (Shipping World, 1948, vol. 18, May 26, p. 517 : [Abstract] Journal of the British Shipbuilding Research Association, 1948, vol. 3, July, p. 328). The Marine Corrosion Sub-Committee of the British Iron and Steel Research Association has made a detailed study of the prevention of corrosion by sea water, and considerable advances are claimed in the development of efficient protective paints. As a result of a series of researches, a number of anti-corrosive compositions have been developed and one of these has been used with satisfaction by the Admiralty. The outer hull plating of the *Pretoria Castle* and *Edinburgh Castle* were painted with compositions and paints to the Sub-Committee's specifications and on docking the *Pretoria Castle*, seven months after fitting out, inspection revealed that 98% of the bottom paint was perfectly intact and unstained by rust.

ANALYSIS

A Note on the Determination of Moisture in Producer Gas. J. Pearson and R. Toye. (Journal of The Iron and Steel Institute, 1948, vol. 159, Aug., pp. 410–414).

Examining Steel by Spot Tests. E. Vrana. (Österreichischer Maschinenmarkt und Eisenhandel, 1947, vol. 2, Aug. 30, pp. 167–172). Drop tests for rapidly determining the following elements in steel are described : Aluminium, chromium, cobalt, copper, manganese, molybdenum, nickel, titanium, tungsten, vanadium, phosphorus, sulphur, and silicon.—R. A. R.

Photoelectric Method of Determining Aluminium in Steel. T. P. Temirenko. (Zavodskaya Laboratoriya, 1947, vol. 13, pp. 621–623 : Chemical Abstracts, 1948, vol. 42, July 10, col. 4486). Dissolve 0·1 g of steel in 10 ml. H_2SO_4, add several drops HNO_3, nearly neutralize with NH_4OH, add 1 ml H_2SO_4, and electrolyse with a mercury cathode. Use a c.d. of 0·16 amp/sq.cm. until the solution does not give a test for Fe^{+3} with $K_3Fe(CN)_6$. The electrolysis lasts for about 20–30 min after which the solution is filtered. Add two drops of methyl orange and neutralize with NH_4OH to the first change in colour. Dilute to 250 ml and withdraw 10 ml to which add 25 ml. of a buffer mixture (equal volumes of 0·1 *N* AcOH and 0·1 *N* NaOAc, 1 ml of aluminium reagent), shake, heat to 90° in 5–8 min (for blank test the interval should be the same) cool rapidly, dilute to 50 ml with the buffer mixture, mix and determine colorimetrically.

Determination of Lead in Steel by the Polarographic Method. Z. S. Mukhina. (Zavodskaya Laboratoriya, 1947, vol. 13, pp. 620–621 : Chemical Abstracts, 1948, vol. 42, July 10, col. 4490). Dissolve 1g of Fe in 20 ml of 6 *N* HCl, add a definite amount of $PbCl_2$, dilute with hot water to 50 ml, withdraw 10 ml of the cooled solution, expel oxygen by passing nitrogen, and then polarograph. This method was used to determine lead in lead-bearing steels. Calculation was made from $\%Pb = Kh/a$ 10 where K is the coefficient calculated from the ratio of $A : H$; A is mg of lead added to the sample of Armco iron, H is the height of the wave in mm corresponding to the amount of lead added, h is the height of the

wave of the test solution, and a is the weight of the sample contained in the aliquot part of the solution transferred to the analyser. Satisfactory results were also obtained by using 12% H_3PO_4. To 5–10 ml. of the standard solution prepared with HCl add 1 ml of 2% NH_4OH-HCl solution and 1·5 ml of H_3PO_4 (d. 1·7), evaporate, dissolve the residue in 20 ml hot water, transfer to an electrolyser, expel oxygen, and polarograph. This method was used to determine lead in steels. Calculation was made with standards and the above equation.

A Thermoelectric Method of Analysing Pig Iron for Silicon. P. D. Korzh. (Zavodskaya Laboratoriya, 1947, vol. 13, pp. 65–68 : Chemical Abstracts, 1948, vol. 42, June 10, col. 3698). Both the silicon and manganese contents influence the thermoelectric coefficient (ϕ) of pig iron. This coefficient is defined by the equation $E/t = \phi$, where E = potential difference between the hot and cold junctions in millivolts, and t = temperature difference between these junctions. To analyse for silicon, a set of standard curves is prepared, in which ϕ is plotted against percentage of silicon for a given manganese content. After the sample has been analysed for manganese by some other method, ϕ is measured and the silicon content read from the appropriate curve. Results agree to within $\pm 2\%$ with chemical analysis.

Photometric Determination of Phosphorus and Silicon in Ferrous Metals. E. I. Fogel'son and F. S. Kazachkova. (Zavodskaya Laboratoriya, 1947, vol. 13, pp. 565–568 : Chemical Abstracts, 1948, vol. 42, July 10, col. 4493). Accurate results were obtainable in the photometric determination of phosphorus and silicon in ferrous metals with an American Fisher electrophotometer and one of Russian manufacture. The Russian apparatus can accommodate a container of specific size only and the container is not suited for rapid analyses ; in addition, the light filters could be improved.

Determination of Silicon with a Steeloscope. N. S. Sventitskii and M. F. Fedorov. (Zavodskaya Laboratoriya, 1947, vol. 13, pp. 626–628 : Chemical Abstracts, 1948, vol. 42 ; July 10, col. 4403). Details are given of an alternating current arc generator for use with steeloscopes for the determination of silicon in steel.

Determination of Copper in Steels with the Aid of the Steeloscope. P. D. Korzh and M. N. Shtutman. (Zavodskaya Laboratoriya, 1947, vol. 13, pp. 190–191 : Chemical Abstracts, 1948, vol. 42, July 10, cols. 4488–4489). A 220-V D.C. arc at 3 amp was used for excitation. The sample acted as anode ; the cathode was an iron rod. After a 2-min pre-arc, the copper line at 5105·6A (A) was compared with the Fe lines at 5097·0 (B), 5098·7 (C), 5107·5 (D), and 5127·4A (E), and the Cu at 5153·2 A (F) with Fe at 5151·9A (G). For Cu = 0·05%, A < B ; for 0·10% Cu, A $\leqslant$ B ; for 0·16% Cu, A $\geqslant$ B and F < G ; for 0·22% Cu, A $\leqslant$ E and F $\leqslant$ G ; for 0·28% Cu, A = E and A < C ; for 0·42% Cu, A > C and F > G ; for 0·50% Cu A > C and A < D ; for 0·57% Cu, A $\leqslant$ D and A > B. Agreement with chemical analysis was excellent.

Potentiometric Determination of Molybdenum and Vanadium in Steels. A. Ríus and M. Martin. (Anales de Física y Química [Madrid], 1947, vol. 43, pp. 897–906 : Chemical Abstracts, 1948, vol. 42, June 20, col. 4090). Molybdenum, after reduction of Mo^{VI} to Mo^{III} by lead amalgam, can be determined in 2M H_2SO_4 solution by titration with ferric alum solution at 100°, oxidizing Mo^{III} to Mo^{IV}. Iron, nickel, cobalt, manganese, zinc, chromium, and copper do not interfere. Lead amalgam reduces V v to a mixture of V^{II} and V^{III}, but zinc amalgam reduces it quantitatively to V^{II}, and Mo^{VI} to Mo^{III}. The V^{II} can then be determined in 2M H_2SO_4 solution by titration in the cold with ferric alum or with NH_4VO_3, oxidizing V^{II} to V^{III}. In a mixture containing both vanadium and molybdenum, the latter is determined in the same solution, by continuing the titration at 100°. Tungsten does not interfere because H_2WO_4 is insoluble in H_2SO_4. The potentiometric curve for the titration of V^{II} with V v has a second inflection point, denoting oxidation to V^{IV}.

Determination of Vanadium in Steels. F. Gómez Ruimonte. (Anales Física y Química (Madrid) 1947, vol. 43, pp. 907–911 : Chemical Abstracts, 1948, vol. 42, June 20, col. 4091). In the usual method for determining chromium and vanadium the end point of the second titration with $KMnO_4$ is difficult to observe when the steel contains more than 5% of chromium, because of the intense green colour of the solution. Better results are obtained by the previous removal of vanadium by the following method : Dissolve 2 g of sample in 50 ml of 6N HCl, oxidize with 10 ml of concentrated HNO_3, and evaporate to 10 ml. If tungsten is present, dilute again to about 50 ml with hot water, remove the WO_3 by filtration and washing, and evaporate once more to 10 ml. Add the solution and rinsings dropwise to 120 ml of boiling 17% NaOH solution in a 500-ml flask with constant agitation. Cool, make up to mark, shake, filter, and wash into a 250-ml flask, using the first 25 ml of filtrate for rinsing the flask. Transfer the 250 ml of solution to a beaker, neutralize with 6N HCl, using methyl orange as indicator, and add 6 ml in excess. Heat until the colour of the indicator nearly disappears, add 50 ml of 20% NH_4OAc solution, and precipitate the vanadium by dropwise addition of 10% $Pb(OAc)_2$ solution to the boiling solution with constant stirring. Collect and wash the precipitate, ignite at 400°, and weigh as $2Pb_2V_2O_7 . PbO$. If molybdenum is also present, dissolve the precipitate in 7·5N HNO_3 on the filter, wash, and determine the vanadium by the usual method which can be done accurately because chromium, iron, and manganese have been removed by boiling with NaOH. Check analyses of steel samples free from vanadium and molybdenum, to which known amounts of vanadium were added, gave satisfactory results, both in the presence and absence of welded molybdenum.

Investigation of the Pyridine Method for the Separation of Some Elements in the Analysis of Alloy Steels. H. P. Samanta and N. B. Sen. (Journal of the Indian Chemical Society, Industrial and News Edition, 1947, vol. 10, pp. 69–72 : Chemical Abstracts, 1948, vol. 42, June 10, col. 3698). In the analysis of alloy steels, treatment with zinc oxide suspension is often used to separate trivalent from bivalent ions, especially in the determination of cobalt. By nearly neutralizing with Na_2Co_3 and finishing with the dropwise addition of pyridine, a good separation was obtained, somewhat more conveniently than with the classic zinc oxide method.

Determination of Small Amounts of Carbon in Steel Evaluation of Low Pressure Combustion Apparatus. J. L. Naughton and H. H. Uhlig. (Analytical Chemistry, 1948, vol. 20, May, pp. 477–480). A comparison of the low-pressure combustion apparatus with standard methods for low carbon (0·012%, 0·014%, 0·069%) and high carbon (0·791%, 0·88%) steels shows good agreement. A series run on the 0·069% carbon steel in samples of 0·005 to 1·1 g indicates that size of sample has no serious effect on the accuracy of this method.—J. P. S.

Method of Determining the State of Carbon in Steel. Yu. A. Klyachko and M. M. Shapiro. (Zavodskaya Laboratoriya, 1948, vol. 14, May, pp. 549–555). [In Russian]. A method is proposed for the separation of free carbon from carbides in the residue formed in the anodic solution of steel. The method is based on the use of a liquid (an aqueous solution of potassium and mercuric iodides) with a density between that of free carbon and that of the carbides. This solution wets the carbide but not the free carbon. The filtration, drying, and grinding of the residue are avoided in the proposed method, which has been thoroughly tested in analyses of carbon, tungsten, and chromium steels which had been subjected to all the usual methods of heat-treatment.—S. K.

The Determination of Carbon in Ferrous Metals, Flue-Dust and Slags. M. S. Kovtun. (Zavodskaya Laboratoriya, 1948, vol. 14, Apr., pp. 487–490). [In Russian]. For the determination of carbon in ferrous metals a flux formed by adding 50 g of copper oxide to 100 g of carbon-free molten lead is recommended since its use enables the combustion of carbon steel, alloy steel, and ferrochromium to be carried out at temperatures of 1000°, 1050°, and 1200° C, respectively. It is noted that the carbon content of samples of cast iron was found to depend on the degree of subdivision, the dust being richest in this element. It has been found advisable, for ferrous metals with 0·2–1% of carbon, to remove the major part of the metal by solution in aqueous chlorides of potassium and copper before carrying out the combustion. For the determination of carbon in flue dust and slags, a method of successive combustions is suggested, with, for samples rich in carbon, the estimation of the residues of this element. S. K.

Use of Alternating-Current Arc with Magnetic Extinguishing as a Light Source in the Spectral Analysis of a Steelometer. P. F. Lokhov. (Zavodskaya Laboratoriya, 1947, vol. 13, pp.

628–630 : Chemical Abstracts, 1948, vol. 42, July 10, col. 4403). Accurate results are obtained in analysing steels for small concentrations of molybdenum, tungsten, nickel, and other elements having weak spectral lines by utilizing as a light source an alternating-current arc with magnetic extinguishing.

Applications of Emission Spectrography in Ferrous Analysis. S. D. Steele. (Journal of the West of Scotland Iron and Steel Institute, 1946–47, vol. 54, pp. 6–27). The author describes modern spark sources, spectrographs, and the photometric methods of determining line intensity. He considers the applications of spectrography in routine analysis, pointing out its great economy in time where a number of elements have to be determined, and discusses the developments in direct photometric measurements of line intensity which have led to the construction of the " quantometer," a direct-reading instrument for the determination of up to eleven elements at once.—J. P. S.

The Influence of Size and Shape of Specimen on the Results of the Spectrum-Analysis of Steels. V. G. Koritskii and T. B. Edneral. (Zavodskaya Laboratoriya, 1948, vol. 14, May, pp. 558–565). [In Russian]. The influence of the shape and mass of the specimen on the results of its spectrum analysis was investigated in a series of experiments in which the following standard conditions for the production of the spectra were observed : (1) Condensed spark with two gaps according to Raiski's scheme ; (2) transformer 110/10,000 V ; (3) capacity 0·01μF ; (4) self-inductance of oscillatory circuit about 20μH ; (5) set spark gap of 3·6 mm ; (6) working gap 3 mm ; (7) auxiliary electrode of carbon, 5·5 mm in dia, ending in a 45°-cone truncated to give a plane 1·5 mm in dia ; and (8) slit 0·015 mm wide. An appreciable systematic error in the determination of the manganese, silicon, chromium, nickel, copper, molybdenum, vanadium, and tungsten in the steel was found to appear when the size of the specimen was reduced below a certain value. This error is due to the disturbance of the spark which occurs when the size of the plane exposed to it becomes less than the size of the " spark stain." The spark stain depends on the geometry of the spark gap, the type of steel being analysed, and the nature of the generator. For the standard conditions used it was found to vary from 9 mm in dia for low-alloy steels to 12 mm in dia for stainless steels. The reduction of the mass of the specimen below a certain value was also found to cause an appreciable systematic error in the analytical results, this limiting value depending on the thermal characteristics of the metal analysed (its thermal capacity and thermal conductivity) and varying to a certain extent for different elements. The experimental results obtained indicate the need for care in the choice of relative dimensions for standard and specimen used in spectrum analysis. With standards having an effective working plane 10–12 mm in dia, specimens with working planes of diameter $\not<$ 10–12 mm can be analysed, but when the plane of the standard is less than 10 mm in dia only those specimens with a plane of exactly the same diameter can be analysed. For conditions other than those used in this investigation, the numerical values of the above limiting dimensions would be different and should be determined experimentally.—S. K.

ECONOMICS AND STATISTICS

Blast-Furnace, Open Hearth and Electric Furnace Statistics. A. Witzig. (Echo des Mines, 1948, May, p. 94). Statistics showing the number of furnaces in operation in France, in January 1938, January 1947, and February 1948, are given.—R. F. F.

Figures of Output, Consumption and Stocks of Coal in Czechoslovakia in 1947. (Bánský Obzor, 1948, vol. 2, Apr. 15, pp. 62–63). [In Czech]. Detailed figures are given for individual collieries in Czechoslovakia, of output, consumption of coal in internal services, in coal processing, and at power stations situated in the immediate proximity of the collieries ; figures for sales and stocks available at the beginning and the end of the year are also given. The output figures are compared with figures of the planned output. The given figures are not always equivalent to the sum for the individual monthly figures published earlier, owing to corrections communicated by the individual collieries after publication of the monthly statistical data.—E. G.

Pig Iron Supply Problems in Hungarian Foundries. B. Körös. (Bányászati és Kohászati Lapok, 1948, vol. 81, May 15, pp. 152–157). [In Hungarian]. Owing to shortages of pig iron Hungarian iron foundries had to use charges containing a very high percentage of scrap, and in 1942 foundries not engaged in war contracts used charges with only 10% to 15%, and even less of pig iron. The quality of the castings obtained deteriorated and the sulphur content was a particularly difficult problem, because it increased with each melt by an average of about 40%. There are six blast-furnaces in Hungary, but their output is required for steel production and little is left over for use in foundries. To obtain good-quality castings it is absolutedly necessary to use charges containing more pig iron and it is suggested that a minimum of 35% to 40% should be made compulsory. As Hungary has to import a part of the scrap used (10,000 to 15,000 tons/annum) it seems reasonable to import pig iron in the first place. It is estimated that about 89,000 tons of scrap and pig iron will be required in the Hungarian foundries this year ; thus about 35,000 tons of pig iron will be required for this purpose if it is desired to use charges with 40% of pig iron. This figure includes the quantities required for special castings, about 7%. Hungarian pig iron is not available in sufficient quantities for this purpose and imported Russian pig iron is expected to fill the gap. Imports started last year ; the Russian pig iron is supplied in 28-kg pigs which break easily and is mainly suitable for use in foundries. The chemical composition is reasonably uniform and the sulphur very low, *i.e.*, 0·020–0·036%. The other elements are : C 3·42–3·76% ; Si 2·60–3·06% ; Mn 0·58–0·79% ; P 0·47–0·70% ; Cu 0·1–0·2% ; and As 0·05–0·1%. The arsenic content causes complications in some cases, but it was found that the trouble was mainly due to sulphur content.

Hematite pig iron is also being imported from Russia ; the chemical composition is favourable and reasonably uniform : C 3·85–4·35% ; Si 3·04–3·22% ; Mn 0·93–0·94% ; S 0·002–0·018% ; and P 0·09–0·11%.

The high arsenic content of many castings seems to indicate that the arsenic content of the scrap is also high, *e.g.*, 0·1%. E. G.

Iron and Steel Specifications in Russia. (Metallurgia, 1948, vol. 37, Apr., pp. 319–320). This is a review of the more important Russian iron and steel specifications. The material is taken from a book on the physical metallurgy and heat-treatment of steels, by Bolkhovitinov published in Moscow in 1946.—A. E. C.

European Iron Economy in Relation to the Marshall Plan. W. Salewski. (Stahl und Eisen, 1948, vol. 68, Jan. 1, pp. 15–19 : Iron and Coal Trades Review, 1948, vol. 156, May 21, pp. 1077–1078).

Reconstruction of the Polish Iron and Steel Industry. E. Dvorzak. (Bánský Obzor, 1948, vol. 2, Apr. 15, pp. 49–56). [In Czech]. The Polish iron and steel industry suffered considerable losses during the war owing to ruthless exploitation of the plant capacity, to the dismantling and carrying away of equipment, and to the use of large amounts of unskilled, mostly forced, labour by the Germans ; the country also served twice as a battlefield during the last war. The total war damage is estimated at about 150 million dollars. In eight ironworks, the whole equipment was lost, in five about half, and in eighteen others a quarter or less. These losses include thirty-six complete rolling mills for sections and tubes ; in the case of the Starachovice Ironworks over 6000 railway wagons were necessary for carrying the plant away. A large proportion of the skilled labour was also lost. The production capacity of coking plant was reduced by 20%, that of blast-furnaces by 30%, that of steelworks by 25%, that of rolling mills by 60% and that of forging plant by 55%. Many coke ovens were cold, several blast-furnaces were suddenly stopped and frozen, and over 55 open-hearth furnaces (80% of the total) were unsuitable for operation. The position was aggravated by lack of spares, raw materials, and workpeople, a large proportion of whom were forcibly drafted and had to be released after the Germans were defeated. Food was also a grave problem and it was so scarce immediately after liberation that employees were given a ration of a calorific value of about 2250 cal per day which had to be shared by the family. Repairs were also hampered by the lack of drawings, which were either destroyed or carried away by the Germans. Sufficient ore was available to permit small-scale operation for several months, but alloying materials were very scarce

and the coal supply was difficult owing to disruption of the transport system, and it was the narrow-gauge railway network which saved the situation. Figures of production capacity are given in thousands of metric tons (including the annexed Western area):

	1938	At Liberation	1.7.1946	Dec. 1946 (planned)
Coke ...	1260	900	1100	1600
Cast Iron ...	1300	400	850	1300
Steel Ingots	1960	650	1500	2000
Sections ...	1380	550	835	1500

Actual production figures are lower than the production capacity. Scrap is in short supply and the proportion of pig iron in the charge of open-hearth furnaces will be increased up to 60% in steelworks situated near blast-furnaces and up to 35% in other steelworks. It is planned to increase the rolling-mill capacity by modernization and the erection of new plant to 1,890,000 tons/annum and that of the coking plant to 1,600,000 tons/annum. Six to eight blast-furnaces of the 21 at present in operation are to be scrapped, and seven modernized; two new blast-furnaces with an output of 600 tons/day are to be built. Steelworks are also being modernized and six open-hearth furnaces are to be built for operating with charges containing 60% hot metal. At the end of 1949 62 open-hearth furnaces and 18 electric furnaces will be in operation. Rolling mills are at present the bottleneck and it is intended to eliminate this by modernizing 38 rolling mills and rebuilding five which were totally destroyed, thus providing an additional capacity of 650,000 tons/annum. It is also planned to build eight new rolling mills with a total capacity of about 315,000 tons/annum; these would include three blooming mills for rolling ingots of up to 11 tons.

The cost of the planned investments is estimated at 90 million dollars, 36 million dollars of which are to be spent on purchases abroad; about 75% of the estimated sum is to be spent in the first two years of the three-year plan (*i.e.*, in 1947 and 1948).—E. G.

BOOK NOTICES

American Society for Metals. "*Metals Handbook* 1948 *Edition*." Edited by Taylor Lyman. 4to, pp. xi + 1444. Illustrated. Cleveland, Ohio 1948: The Society. (Price $15.00).

The title "Metals Handbook" is somewhat misleading in so far as the 1948 edition is concerned. A much better title would be "Metals Encyclopaedia" because it is an encyclopaedia, and a very good encyclopaedia. It consists of more than 1200 pages covering practically every aspect of the properties, treatment, and uses of all known metals, and is not only an adequately indexed reference book but also a book that one can browse through and enjoy reading.

On the front fly leaf there is a condensed table of contents, and an index of important reference tables. Inside the back cover there is a guide to the non-ferrous data sheets. The complete table of contents is given on pages vii to xi and lists the titles of all the 803 articles in the Handbook.

The index comprises the last 64 pages in the book and is easily found because it is printed on green paper.

The enormous range covered can be best illustrated by reference to two of the articles. There is an excellent article on the open-hearth process of steelmaking by A. H. Hurtey, jun., in which there is a table showing the division of time for making a 200-ton open-hearth heat. At the other end of the scale Sam Tour in an article devoted to jewellery casting, describes an old jewellery practice suitable for making men's heavy rings which utilizes a mould from cuttlefish shell. The shell is carefully sectioned through the middle and the halves are smoothed by rubbing them on fine sandpaper. The object to be reproduced is then pressed gently into one plane surface just far enough to seat it firmly, whereupon the other plane surface is pressed against the other half of the object. The mould is then opened gently and the gate routed in with any convenient hand tool.

Many of the articles have been contributed by well-known experts, and the remainder are the work of various sub-committees of the American Society for Metals.

The book is divided into four main divisions, *viz.*: General Section, Ferrous Metals, Non-Ferrous Metals, and Constitution of Alloys. Under these four headings are listed the 37 sections of the Handbook, of which reference will be made to two taken at random, namely, Case-Hardening and Precious Metals.

Twenty-one pages are devoted to case-hardening and there are ten articles by well known experts dealing with carburizing methods; mechanism of carburization; selection of carburizing steel, case depth, and heat-treatment; measurement of case depth; pack carburizing; gas carburizing; liquid carburizing; selective carburizing; cyaniding; carbo-nitriding and nitriding. The articles dealing with the mechanism of carburization, gas carburizing, and liquid carburizing are particularly good.

The Precious Metals section has been prepared under the chairmanship of E. M. Wise of the International Nickel Company Inc., and opens with an interesting article by Mr. Wise himself on the precious metals industry. Then follow articles dealing with the melting of precious metals; jewellery casting; dental casting; working of platinum metals; forming and cutting of gold alloys; joining and cladding precious metals; electroplating with precious metals; polishing precious metals; and metallography of precious metals. Finally come sections detailing properties of silver and silver alloys; gold and gold alloys; platinum and platinum alloys; palladium and palladium alloys; and of the metals iridium, osmium, rhodium, and ruthenium.

A particularly interesting and useful section of the Handbook is that dealing with the constitution of alloys which has been prepared under the direction of the Phase Diagram Committee of the American Society for Metals; it gives 280 phase diagrams.

Finally, there is an advertising section printed on buff coloured paper and covering 112 pages. This is not obtrusive but is easily found and worth perusal.

J. Ferdinand Kayser

F.I.A.T. Review of German Science, 1939–1946. "*Physics of Solids*"—Part I. Senior Author Georg Joos. 8vo, pp. 228. Berlin, 1947: Office of Military Government for Germany. Field Information Agencies, Technical. [In German].

F.I.A.T, the Field Information Agency (Technical) of the British, French, and U.S. Military Governments of Germany, has arranged for the scientific work done in Germany during the war to be reviewed by panels of German scientists. It is intended that the Review shall be published commercially in Germany: copies of the present limited printing are deposited in the principal libraries. The Review is to occupy 86 volumes, of which 16 are devoted to physics and three to metallurgy. This volume is the first of two volumes on the physics of solids, and is concerned with the structure, mechanical properties, and thermodynamics of solids. The second part treats optical, magnetic, and electrical properties.

The aim of the writers has been to give a very brief summary of work done, with plentiful references. The first article, by R. Glocker, on methods of structural investigation, describes the beautiful investigation by Brill, Grimm, Hermann, and Peters (1939) of the electron distribution in solids. By Fourier analysis of X-ray photographs, they were able to confirm the basic results of the electronic theory of cohesion: in the ionic lattice of NaCl the space between the ions is almost free from charge, while in the diamond lattice the minimum electron density on the line joining two neighbouring atoms is much larger, and the existence of a covalent bond is directly demonstrated. In magnesium there is an almost uniform background of charge with a density of 0·1 electrons/Å^3, which corresponds to a uniform distribution of nearly 2 electrons/atom. The work of Dehlinger and Kochendörfer on the influence of particle size and lattice distortion in line-broadening and that of Glocker and others on X-ray measurements of internal stresses are discussed. The section on crystals and crystal growth by D. Kossel discusses surface forces

and growth, the imperfections of real crystals, and the relation between finely crystalline and amorphous solids. M. Mehmel summarizes research on the crystal chemistry of non-metallic substances. U. Dehlinger discusses the heat of formation and the nature of the binding forces in alloys, and the kinetics of precipitation and allotropic transformation. There follow articles by A. Smekal on glass and plastics, and by H. Raether on the structure of the films formed by evaporation or oxidation. Raether also contributes an article on the structure of cold-worked surfaces. From the transmission electron diffraction pattern of thin films of gold and and copper he deduces that the line-broadening produced by cold work is genuine and not a spurious refraction effect. To decide whether the polished layer is microcrystalline or truly amorphous, he considers bismuth, antimony, and selenium, which possess amorphous modifications or liquids with very different structures from the crystalline solids. In every case the interference from the polish layer corresponds to that of the finely fragmented crystalline solid and not to that of the amorphous phase, and he assumes from this that polish layers in general are microcrystalline.

The second chapter, on mechanical properties, begins with a paper by W. Köster on the measurement of elastic constants, and the influence of composition, working, and temperature on these constants. He has prepared an alloy of iron and nickel which has a practically constant Young's modulus over a temperature range of 250°. A. Kochendörfer writes on the plasticity of crystalline solids, a subject on which he and Dehlinger have developed a theory in which the strength is due mainly to the difficulty of forming dislocations. He describes his own important ideas on the relation between the strengths of single crystals and of polycrystals, and also the experiments of Smekal's school, which show that the force required to make a fine scratch on glass corresponds closely to the theoretical cohesive strength. He also suggests a theoretical connection between the creep and fatigue limits. Th. Pöschl and H. Böckle discuss the theory and application of the Brinell hardness test, and describe the Hanemann microhardness tester (Zeiss) which contains a Vickers pyramid diamond mounted on the objective of the observing microscope. Impressions 8μ across can be used. There is a remarkable observation by E. O. Bernhardt that the hardness of some globular graphites is almost as great as that of diamond. The next paper, by E. Hiedemann, deals with ultrasonics and the propagation of waves in solids. The chapter ends with a paper by K. Bennewitz on internal friction in solids, including problems in which there may be a permanent plastic set.

The chapter on the thermodynamics of solids begins with a section by K. Wirtz on thermal expansion. For alkali halides the thermal expansion observed with an interferometer exceeds that of the lattic observed by X-rays. Since aluminium observed in the same apparatus showed no discrepancy, the excess expansion of the crystal is attributed to the formation of Schottky lattice vacancies in the halides. The thermal expansion of crystalline silicon is anomalous at low temperatures; a minimum in the thermal expansion of chromium near room temperature corresponds to a minimum in the electrical resistance, but there is no anomaly in the thermal conductivity. K. Clusius reports on the specific heats of solids, covering apparatus; theory of specific heats at low temperatures; specific heats of metals and the anomalous electronic specific heat of palladium, which is proportional to the paramagnetism; specific heats of iron and steel to high temperatures, and of chromium in the region of the anomaly; specific heats of alkali halides at low temperatures; specific heat, heats of transformation, and absolute entropy of GeH_4, and of chlorides, bromides, and iodides of hydrogen and deuterium; and miscellaneous specific-heat measurements, including various types of ccal. The next paper by G. Hettner on phase changes deals with the detection of phase changes, order-disorder transformations, onset of free rotation, transformations in CH_4, CH_3D and CD_4, and miscellaneous transformations. W. Meissner and G. O. Schubert treat problems of heat conduction, including experimental methods, results for metals, coal and coke, synthetic materials, etc., the thermodynamics of irreversible processes, solutions of the equation of heat conduction, and technical applications. The last paper, by W. Braunbek, discusses diffusion in solids. Among other methods he discusses observation of the diffusion of impurities in luminescent solids by observing the change they produce in the luminescence. Diffusion of gases in solids, interdiffusion of metals, separation of the isotopes of copper by diffusion through Ag_2S (5·5% enrichment with a yield of 20 mg), practical applications, including the diffusion of chromium into steel and the decarbonization of steel, and the theory of diffusion, are treated.

There is some duplication, but this can hardly be avoided in a collective work, and the volume is a most useful guide to the rather inaccessible literature of German wartime research.—F. R. N. Nabarro.

Jacquet P. A. "*Le Polissage Électrolytique des Surfaces Métalliques et Ses Applications.*" Vol. I—"*Aluminium, Magnesium, Alliages legers.*" La. 8vo, pp. xiii + 359. Paris, 1948. Editions Métaux. (Price 3600 frs.).

Dr. Jacquet was the discoverer of electropolishing and it is therefore most fitting that he should produce the first comprehensive treatise on this subject, which already has an immense literature. In this, the first of two promised volumes, only aluminium, magnesium, and light alloys are dealt with. Electropolishing is of course an electrochemical method of producing a microscopically smooth and hence highly reflecting surface on metals. In the preparation of metal specimens for physical or metallographic examination of the surface, its great importance is not so much the reduction of the physical labour of preparation but the provision of a smooth surface truly representative of the bulk of the material and free from the structural changes and surface contamination unavoidable in mechanical methods of preparation. The major part of this volume is devoted to this type of application. First the various successful methods of treatment, not only of Jacquet and his collaborators, but also of other workers, are described in considerable detail (71 pages). Since these are used as laboratory tools they justify the careful and exacting technique and somewhat unpleasant solutions recommended. Jacquet's own method, and some of the others, is based on a mixture of perchloric acid and a readily oxidized organic liquid. There is, however, no discussion of the reason for the efficacy of this somewhat incompatible and even dangerous mixture, although there is a section discussing the explosive risk. One is left wondering by what process of trial or reasoning such a composition was evolved. Indeed there is nowhere in this volume a general discussion of the principles on which the process rests and no generalization by which the widely differing types of advocated solutions can be linked up.

The next section (72 pages) is devoted to the use of this method of surface preparation, to studies of reflectivity, diffraction, etc., with visible light, X-rays, and electron waves, also studies of oxide films, glide planes, and etch pits, and the direct realization of the true electrochemical properties of aluminium metal.

After this come 150 pages devoted to the elucidation of the microstructure of light alloys using the powerful new tool; these are illustrated with over 200 superb photomicrographs.

The industrial applications (to aluminium and light alloys) are dealt with in 25 pages. This, at first sight, cursory treatment is justified because in spite of its name, electropolishing produces surfaces, which, though brilliant and microscopically smooth over short distances, are in general not sufficiently optically true over macroscopic areas to yield unblurred images when viewed from a convenient distance by the unaided vision. Thus the process is not a direct substitute for the laborious, costly, and dirty methods of mechanical polishing; use of the term "electrobrightening" in this connection avoids raising undue hopes. The two commercially successful processes (Brystal and Alzak) are thus almost entirely limited to the electrobrightening of lamp reflectors and similar articles, which are subsequently anodized without detriment, and indeed with preservation of, their enhanced reflectivity.

The volume closes with a selected bibliography of some 200 items, and an author, but not a subject, index.

It will be appreciated that, whilst there is nothing in

this volume, except the promise of the second volume, of direct interest to the steel metallurgist, nevertheless it will be of considerable general and scientific value to him. To the technical metal polisher it will be disappointing.

The paper, printing, illustrations, and binding of this quarto volume, with its 1½-inch margins on each side of the letterpress, and prodigal use of space, will be the despair of British technical book publishers still bound down to authorized economy standards.—G. E. GARDAM.

GUILLET, L. "*Structures et Propriétés des Alliages Métalliques.*" 8vo, pp. 206. Illustrated. Paris, 1948: Dunod (Price 680 frs.).

This is an elementary textbook of metallography on conventional lines. It is clearly written and covers the field required by students, and should be useful in French universities and colleges. There are no special features which would appeal to English students, and the very numerous errors in the footnotes are irritating. The diagrams, mostly from well-known sources, are clearly printed. X-ray methods are included, but it would have been well to have given more details of the manipulation of the microscope, which still remains the principal tool of the metallographer, in spite of the progress made in applying other methods of investigation.—C. H. DESCH.

SAUNDERS, B. "*Heat Treatment in Forging Steel.*" A Manual on the Application of Science to the Heating, Cooling, and Forging of Steel. 8vo, pp. viii + 117. Illustrated. London, 1947: The Technical Press, Ltd. (Price 6*s*.).

This is a reissue of a book that was first published in 1930. The publishers have felt justified in reissuing it in the belief that it will be of value to readers. The author's object is to supply some notes and examples, expressed as plainly as possible, of the science of smithing. Attention is drawn to the important of temperature control, and the instruments in use for this purpose are illustrated and described.—R. E.

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